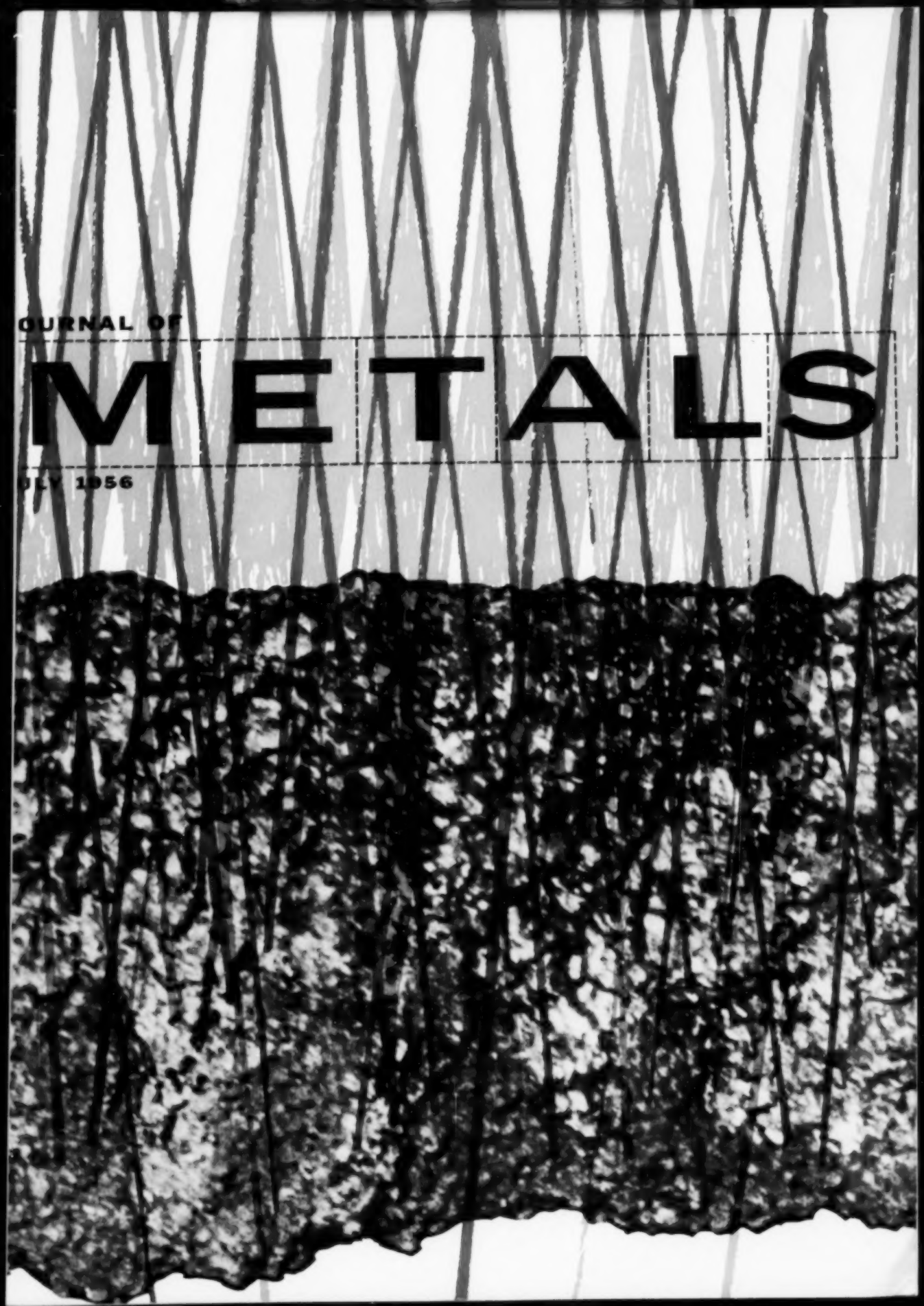
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COVER

Artist Harry Brocke chooses this colorful representation of the sintering process for this issue devoted to summarizing developments of ferrous sintering. The history of the Dwight-Lloyd technique is recounted on page 828, and subsequent pages are devoted to descriptions of recently constructed ferrous sintering plants.

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THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

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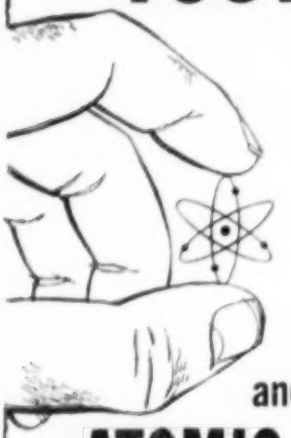

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The Analysis of Minerals and Ores of the Rarer Elements, A Manual for Analytical Chemists, Metallurgists, and Advanced Students, by the late W. R. Schoeller and A. R. Powell, *Hafner Publishing Co.*, $9.75, 408 pp., 3rd edition, 1955.—This third edition has been extensively revised by A. R. Powell, research adviser, Johnson, Matthey & Co. Ltd., London. This revision is particularly marked in the chapters on the rare-earth metals—thorium, germanium, niobium, and tantalum. The chapters on scandium, uranium, rhenium, and the platinum metals have been almost completely rewritten and contain many new analytical methods.

ASTM Standards on Light Metals and Alloys, Cast and Wrought, *ASTM*. $3.50, 276 pp., 1956.—This third edition sponsored by Committee B-7 includes specifications for die castings and aluminum wire, and cable for electrical purposes. Methods of testing and chemical analysis of metals, as well as specifications for aluminum, aluminum alloy arc-welding electrodes, and brazing filler metal have been included. The standards include ingots, castings, bars, rods, wire, forgings, pipe and tube, sheet and plate, wrought products for electrical purposes, and general methods of test. ASTM codification system for light metals and alloys is also offered.

An Encyclopedia of the Chemical Process Industries, by Jeffrey R. Stewart, *Chemical Publishing Co. Inc.*, $12.00, 820 pp., 1956.—Written by the former special assistant to the Quartermaster General, Dept. of Defense, this book treats the practical aspects of raw materials, processes, equipment, and finished products of the chemical process industries. Included are definitions of technical and scientific terms frequently encountered in these industries. As it also lists the manufacturers and uses of thousands of trade-name products, it should serve as a buyer's guide.

The College Blue Book, edited and published by *Christian E. Burckel*, P.O. Box 311, Yonkers-on-Hudson, N. Y., $12.00, 688 pp., 1956.—A standard work of reference of higher education in the U. S., this eighth edition replaces the 1953 edition. The volume supplies information about every institution of higher learning requiring graduation from a secondary school for admission. It includes a section on engineering schools by W. Leighton Collins, Secretary ASEE, and lists the engineering curricular accredited by ECPD. Other sections of particular interest to engineers are devoted to technical institutions and to scholarships, foundations and research.

English-German and German-English Dictionary for the Iron and Steel Industry, by Eduard L. Köhler, *Springer-Verlag*, $6.65, 162 pp., 1955.—Only directly pertinent terms have been included and those from related fields, i.e., plant economy, omitted, so that all the divisions of the iron and steel industry could be included. The terminology of the most recent developments are covered and equivalents are frequently explained by definitions which indicate the field of usage. American spellings are used throughout.

ASME Handbook: Engineering Tables, edited by Jesse Huckert, *McGraw Hill Book Co., Inc.*, $12.00, 704 pp., 1956.—This reference book, sponsored by the American Society of Mechanical Engineers is one of four which comprise the complete **ASME Handbook.** It contains many tables of design standards not generally found in handbooks, such as the table of involute functions, methods for finding loads on ball bearings and information on bars and tubes suitable for reworking into gears, levers, shafts or screws. Conveniently arranged in 15 sections the book covers such subjects as bar stock and shafting; bearings; bevel, and worm gears; keys; bolts; springs; aircraft and mechanical tubing; and electrical motors. The author, who is professor of mechanical engineering at Ohio State University, was formerly associate editor of *Product Engineering*.

Exploration for Nuclear Raw Materials, by Robert D. Nininger, *D. Van Nostrand Co. Inc.*, $7.50, 1956.—The author, AEC assistant director for exploration, has compiled highlights of 130 papers from 26 countries. The methods of prospecting for nuclear metals are developed in logical sequence from nuclear geology to geochemical and geophysical techniques. Included are details of surface, sub-surface and aerial radiometric methods with special emphasis on their uses in the U. S., Canada, Brazil and U.S.S.R.

Legal Problems in Engineering, by Melvin Nord, *John Wiley & Sons Inc.*, $7.50, 391 pp., 1956.—The author, who is an engineer as well as a practicing lawyer, attempts to help engineers avoid legal problems before they arise. Dr. Nord emphasizes legal problems and uses cases to illustrate his discussion. By condensing cases and keeping legal jargon to a minimum, the author offers assistance to the nonlawyer, seeking a reliable, concise survey of the field.

The Metallurgy of Zirconium, edited by Benjamin Lustman and Frank Kerze, Jr., *McGraw-Hill Book Co. Inc.*, **Div. VII, vol. 4, National Nuclear Energy Series**, $10.00, 776 pp., 1956.—More than 50 specialists have prepared this book which is a complete report on the present state of zirconium, covering occurrence, reduction, refining, properties, fabrication, and uses. It will be of interest to research and operating engineers concerned with plants handling zirconium and its alloys. An outgrowth of the AEC and the U.S. Navy's effort to use atomic energy for marine propulsion, the book is particularly strong in its treatment of such topics as corrosion properties of zirconium, fabrication techniques, and the iodide method of refining.

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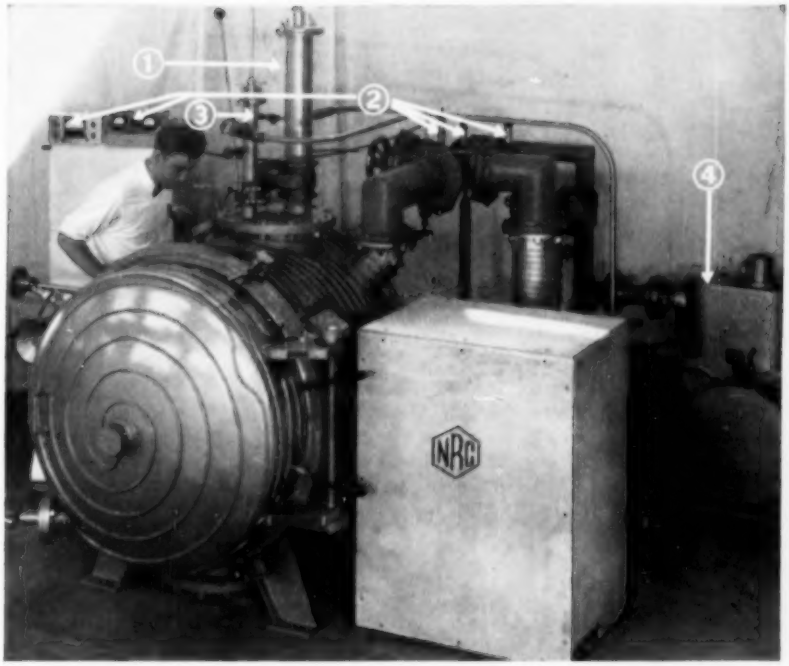

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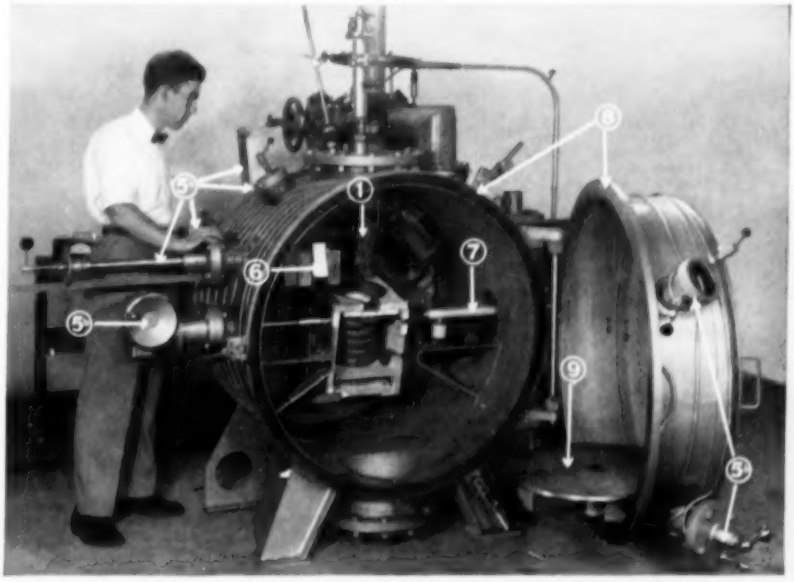

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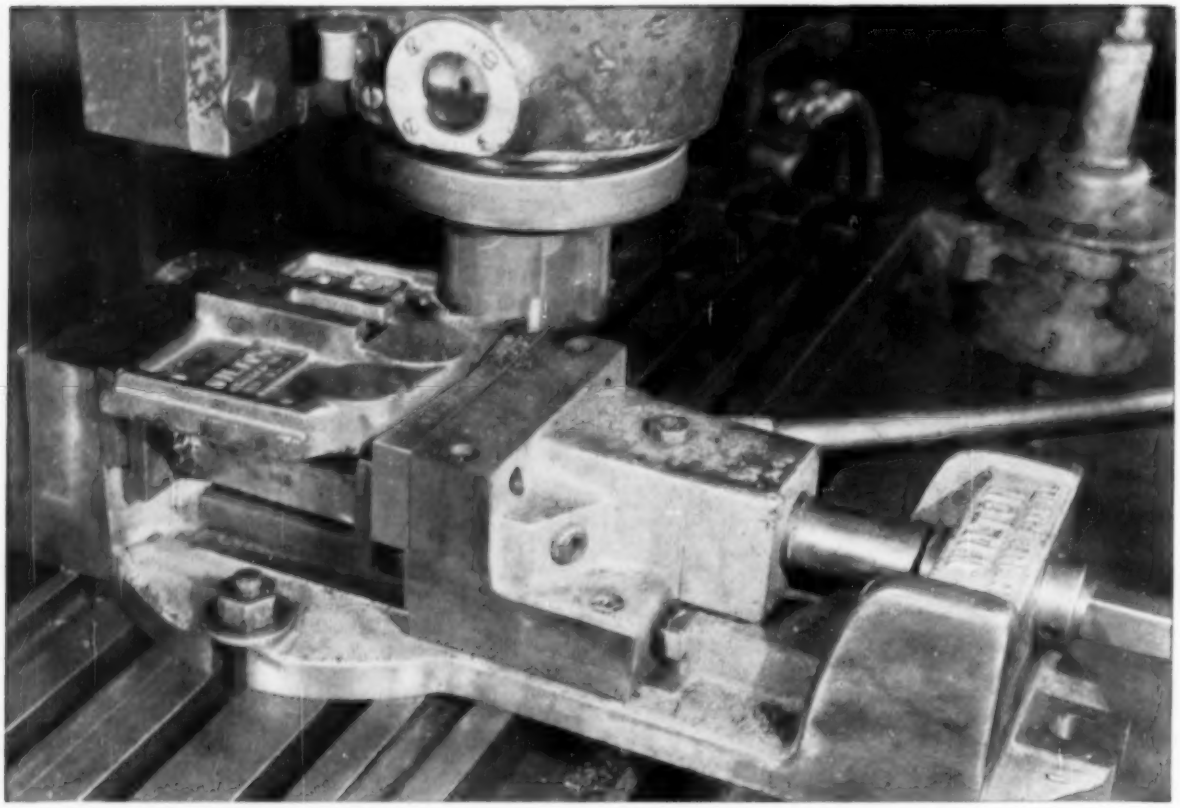

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REPORTER

Continuing Expansion of Sintering Facilities

In Canada, Algoma Steel Corp. is raising sinter capacity from 1.4 to 1.8 million tons by 1957. At present about half of Algoma's self-fluxing sinter is exported to the U. S. . . . A 5000 tpd sintering unit is being added to U. S. Steel Corp.'s Ohio works. . . . U. S. Steel Corp. has also begun construction of a three unit sintering plant at Houseville, near Butler, Pa. The 15,000 tpd plant will supply sinter to plants at Duquesne, Clairton, Homestead, and Braddock, Pa. A plant of similar capacity is also being laid down for U. S. Steel Corp. at Gary, Ind. Both plants, with 8-ft, 3-in. wide Lurgi sintering strands, are being built by the Dravo Corp.

Fairless Works Adding Third Blast Furnace

At Morrisville, Pa., the U. S. Steel Corp. will erect a new blast furnace identical to the two already in production. The construction contract for the 1600 tpd furnace has been awarded to Arthur G. McKee & Co. with completion scheduled for next year. Already under construction is a sintering unit for agglomerating ore fines.

Philadelphia to Have Ore Agglomeration Plant

With the completion of Pennsylvania Railroad's $12 million ore pier in 1954, Philadelphia has become a major iron ore importing center, handling almost 7 million tons in 1955. In close proximity to the pier at Greenwich Point, construction work is underway on a plant for screening iron ore and pressing the fines into blocks for use in open hearth practice. The plant, being built for the Blocked Iron Corp. of America, will commence operation of the first unit in August.

Rising Consumption in European Steel Community

Steel consumption in the European Steel Community has risen by 70 pct during the past five years to 271 kg per capita, still considerably below the 620 kg in the U. S. Within the Community, Germany's 1955 per capita consumption was the highest, 438. . . During the first five months of 1956 the Community's steel production continued to climb and is currently running about 8 pct above the comparable period of the record-breaking 1955.

Russia Too!

Direct introduction of oxygen into the open hearth bath by tubes inserted through furnace doors has been used at the Serp-i-Molet works in Moscow. At Zaporozh-stal, a 200-ton fixed furnace has used oxygen lance through the roof, while at Azov-stal, oxygen has been introduced to a 350-ton tilting furnace through the back wall. Further experiments are expected, for the current Five-Year Plan calls for construction of oxygen plants at steelworks. . . Russians are also interested in the new oxygen converter process for their medium phosphorus iron. Their 1955 experiments were so successful that the process is to be installed at two plants in the Ukraine.

Nose wheel landing strut piston is inspected with a G-E OX-250. Power of 250,000 volts permits short exposure times without disassembly.

G-E x-ray units speed aircraft inspection

X-RAY inspection-without-disassembly is routine preventive maintenance at the St. Paul, Minn., overhaul depot of Northwest Orient Airlines. Records show their two General Electric x-ray units not only save time and money, but also contribute to longer parts service life. Oil cooler radiator overhaul, inspection of nose wheel strut piston . . . examination of prop blades . . . welds — these are typical jobs now assigned to x-ray.

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Have you considered x-ray for inspection and quality control? Ask your G-E x-ray representative for facts on this complete line of x-ray equipment. Choice of 25 models in every practical voltage range. Or write X-Ray Department, General Electric Company, Milwaukee 1, Wisconsin Dept. AY-74.

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Engineering Enrollment

by W. B. Plank

MINERAL engineering student enrollment in U. S. and Canadian schools for 1955-1956 is 11,408, an increase of 11 pct more than last year. The undergraduate and graduate engineering students in both countries number 251,765, or 13 pct more than last year. These figures have been released by the American Society for Engineering Education.[1]

The largest increase in the undergraduate mineral engineering group in the U. S. is in ceramic engineering, with 676 students, or 31 pct more than last year. The next largest increase is in mining which shows a gain of 13.5 pct, compared with a gain of 13.1 pct in the total undergraduate enrollment in all engineering schools approved by the Engineers' Council for Professional Development. This figure is 190,355, which approaches the prewar figure. Metallurgical undergraduates increased 13 pct; geological, 5 pct; and petroleum, 2.5 pct. This year the figures for geophysics are combined with those for geological engineering, making a total of 1610 undergraduates, of which the geophysicists comprise about one seventh.

Again this year, the metallurgical freshman class is smaller than the three other classes. This may be accounted for by segregation of the metallurgicals from other students in the first year, as with chemicals, a condition that exists at a number of engineering schools. A partial answer to this situation[2] is that last year's entering class of metallurgicals, 451 students, now numbers 619 in its second year.

Among the 23,733 first degrees conferred last June by U. S. and Canadian schools, 1361 were to mineral engineers in the U. S. Of these, 183 were in mining, 436 in metallurgy, 450 in petroleum, 86 in ceramics, and 206 in geology.

On the whole, engineering enrollment is assuming the normal increase it was following before World War II, although the effect of the decreased college population during the 1930's has not been entirely erased. The comparative increase shows encouraging response to the publicizing, by national agencies, of the unfavorable world situation of the U. S. with respect to the need for more engineers and scientists for modern industries and national defense.

The present student response to the opportunities in all branches of mineral engineering, on which an adequate supply of mineral raw materials vitally depends, should encourage mineral engineering educators and administrators of their institutions. However, because of the rapid change and increase of the mineral sciences, educators should constantly reinterpret the objectives of their fields to the young men of college age.

W. B. PLANK, Member AIME, deceased, was Emeritus Professor of Mining, Lafayette College, and Consulting Engineer, Morgantown, Pa.

Engineering Student Enrollment Comparison
Undergraduates and Graduates

Courses	1950 to 1951*	1951 to 1952*	1952 to 1953	1953 to 1954*	1954 to 1955*	1955 to 1956**
Mining	1,607	1,338	1,168	1,089	1,118	1,239
Metallurgical	3,511	2,998	3,085	3,159	3,050	3,421
Petroleum	3,321	2,690	3,446	3,772	4,110	4,195
Ceramic	886	667	579	557	626	767
Geological	1,349	1,027	1,173	1,249	1,370	1,786
Total Mineral Engineers, U. S.	10,674	8,727	9,453	9,826	10,274	11,408
Chemical	16,323	14,079	15,026	15,858	17,097	18,562
Civil	24,505	21,801	22,393	23,075	23,799	26,060
Mechanical	35,347	30,420	32,791	35,097	38,587	43,166
Electrical	34,105	29,966	32,848	37,633	43,305	52,316
Industrial	?	6,578	6,843	6,786	7,168	7,747
Other Engineers	59,308	36,123	39,364	43,557	47,224	52,664
Total Engrs., ECPD	?	147,694	158,518	171,832	187,454	211,923
Other U. S. Engrs.		17,943	18,031	21,501	26,960	31,467
Total U. S. Engrs.	180,262	165,637	176,549	193,333	214,414	243,390
Canadian Engrs.	4,285	5,433	5,812	6,894	7,764	8,375
Total Engrs. U. S. and Canada	184,547	171,070	182,361	200,227	222,178	251,765

* MINING ENGINEERING, April 1955.
** *Journal of Engineering Education*, February 1956.

Bachelor Degrees, All Engineers, U. S. and Canada

1942 to 1943	14,714	1949 to 1950	54,441
1943 to 1944	11,155	1950 to 1951	43,762
1944 to 1945	4,724	1951 to 1952	31,660
1945 to 1946	8,219	1952 to 1953	25,332
1946 to 1947	19,272	1953 to 1954	23,272
1947 to 1948	30,018	1954 to 1955	23,733*
1948 to 1949	46,934	1955 to 1956	35,976**

* U. S., 22,589 and Canada, 1,144.
** Estimated—Fourth and Fifth Year Students.

References

[1] W. A. Jarecz and H. H. Armby: Engineering Enrollments and Degrees, 1955. *Journal of Engineering Education*, February 1956.
[2] W. B. Plank: Mineral Engineering Student Enrollment Passes 10,000. MINING ENGINEERING, April 1955, p. 357.

Undergraduate and Graduate Engineering Enrollment in 216 Schools in the U. S. and Canada, 1955 to 1956*

Schools	Courses	Fresh.	Soph.	Junior	Senior	5th Yr. & Others	Total Undergards	Grad. Students	Grand Total
31	Mining	343	299	267	253	24	1,186	53	1,239
48	Metallurgical	538	619	638	573	241	2,609	812	3,421
22	Pet. and N. Gas	1,032	1,125	965	726	182	4,030	165	4,195
14	Ceramic	208	197	145	113	13	676	91	767
30	Geological and Geophysical	444	500	360	287	9	1,610	176	1,786
	Total Mineral Engineers U.S.	2,565	2,740	2,375	1,962	469	10,111	1,297	11,408
106	Chemical	4,590	4,270	3,499	2,771	988	16,098	2,464	18,562
136	Civil	6,166	5,950	5,097	4,555	2,016	23,784	2,276	26,060
132	Mechanical	9,795	10,040	8,716	6,853	4,287	39,691	3,475	43,166
136	Electrical	12,173	12,374	9,789	6,950	4,549	45,835	6,481	52,316
63	Industrial	966	1,341	1,526	1,615	543	5,991	1,756	7,747
	Other Engrs.	5,364	4,707	3,538	3,104	1,353	18,066	3,258	21,324
94	Unclassified	19,522	2,353	156	59	8,689	30,779	561	31,340
149	Total Engrs. E.C.P.D. Schools	61,141	43,775	34,696	27,889	22,874	190,355	21,568	211,923
67	Other U.S. Engr. Schools	11,684	7,066	4,681	3,431	4,231	31,093	374	31,467
216	Total U.S. Engineers	72,825	50,841	39,377	31,390	27,105	221,448	21,942	243,390
10	Total Canadian Engineers	2,662	2,042	1,524	1,587	282	8,097	278	8,375
226	Grand Total, U.S. and Canada	75,487	52,883	40,901	32,887	27,387	229,545	22,220	251,765

* From *Journal of Engineering Education*, February 1956.

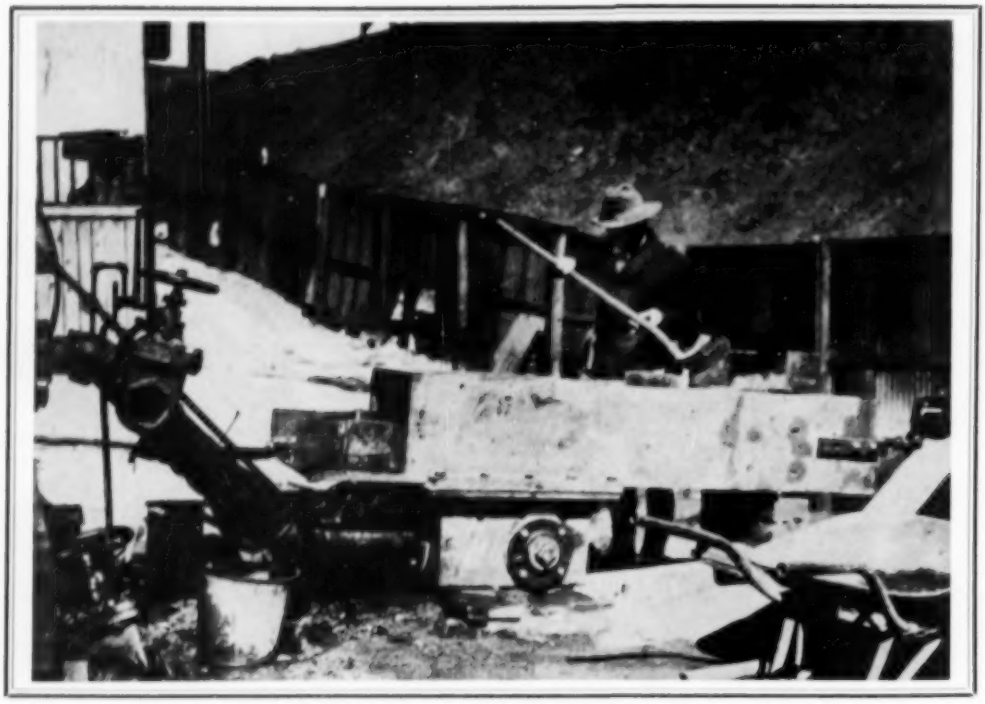

Fig. 1—R. L. Lloyd charging the first successful continuous sintering machine.

Development of the Dwight-Lloyd Sintering Process

by H. E. Rowen

As high grade iron ore deposits dwindle and costs rise, sintering becomes more and more important. The steel industry is now faced with beneficiation problems once peculiar to nonferrous work. Succeeding articles describe recent advances in this field. Here is presented the first chapter of a history still being written.

SINTERING, for the purpose of this discussion, is defined as *the art of burning a solid fuel with 90 to 95 pct ash content*. Considering problems of combustion involved in keeping a home furnace burning properly with 12 to 20 pct ash in the fuel, it is evident that combustion represented by the definition has to be right or nothing is going to happen.

Our forefathers discovered that many ores containing metals essential to the steadily rising standards of living were associated with elements such as sulfur and arsenic, which had to be removed before refining. This they did by utilizing the solid fuels they had available—the sulfur or other readily combustible components—to create roasting heat.

One of the early methods is demonstrated in an old woodcut of the *roast heap*, Fig. 2. Ore was broken into chunks, piled on an ignition layer of solid fuel—logs in this case—and then ignited. The sulfide ores roasted as the sulfur burned out. As shown in the background, the countryside was rapidly denuded of all vegetation by the great quantities of SO_2 and other liberated gases.

H. E. ROWEN is General Manager, Dwight-Lloyd Div., McDowell Co. Inc., Cleveland.

Fig. 2—The roast heap, from an early woodcut.

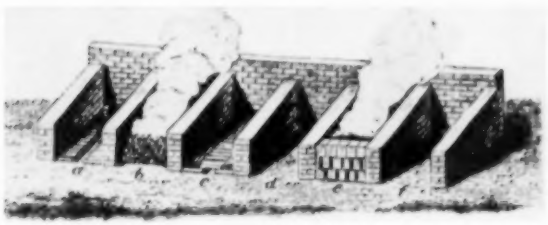

Fig. 3—The roasting stall, a decided improvement.

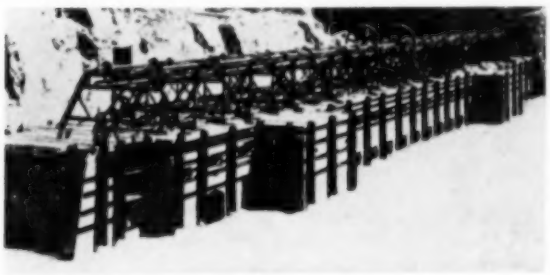

Fig. 4—Mechanically rabbled reverberatory furnace.

Because burning of heaps was erratic, time-consuming, and uncontrolled as to product, man's inventiveness soon created the *roasting stall*. At *A*, Fig. 3, the first layer of ignition logs is laid with spaces between them and a channel between the ends for passage of air. At *C* the second layer of logs is laid crosswise and endwise to support the bed of ore, and at one of the stalls, *B*, ignition has been accomplished. At *E* the draft control has been set up to prevent uncontrolled burning. Although this was a start toward continuous production, like the earlier method it destroyed the countryside. The famous Tennessee Copper Basin still shows results of these older operations.

Most of the ores treated were copper and lead ores, and not all the sulfur could be removed if they were to be left in condition for further smelting. With constant efforts to control the process and composition of the product and to make the roasting continuous, modern sintering practice gradually evolved.

Early vertical kilns for coarse ore were merely refractory columns with top feeds, controlled natural side drafts, and exit stacks. The hand-rabbled reverberatory furnaces later used for roasting required a large ventilation area for the men who were working.

The old revolving cylinder roasters need little description. Loaded from hoppers near the center, they were operated like peanut roasters and then dumped out.

Hand rabbling of the reverberatory soon gave way to a mechanical rabbling furnace, Fig. 4. Comparatively high shaft furnaces with deflector plates, as well as the oxidizing furnace, were used. These were followed by continuous roasting in round hearth rabbling furnaces, of which the six-hearth Herreshoff furnace is typical.

All these devices left something to be desired. The product was not in a proper physical state for furnace charging, great quantities of flue dust were produced, and considerable metal values were lost. It was obvious that the product had to be agglomerated in some manner.

An early answer was provided by the Huntingdon and Heberlein blast roasting pot, Fig. 5. The ore, bedded down in a pot with a perforated bottom plate, was bottom-fired, and a cover with offtake pipe was fitted on. After a long period of blowing, a large agglomerated cake was formed, and everything appeared to be solved. The trouble began when the roasting pot was dumped, Fig. 6. To make matters worse, dumping was followed by a heavy hand-breaking operation, Fig. 7.

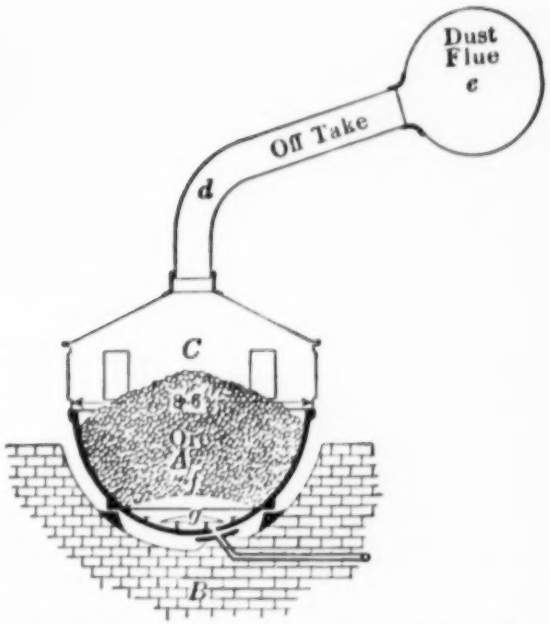

Fig. 5—Diagram of the Huntingdon and Heberlein blast roasting pot.

This was the type of equipment A. S. Dwight and R. L. Lloyd had to work with when they were superintendent and metallurgist, respectively, of the Greene Consolidated Smelter in Cananea, Mexico. An obvious solution was to develop some continuous method of production and to make the cake of agglomerated material thin enough to be broken.

A long series of experiments started, resulting in the general developments shown in Fig. 8. *A* is the Huntingdon-Heberlein principle of blowing upward through a deep bed of ore. At *B* there is a reversal of air flow. Neither method was suitable,

Fig. 7—Breaking up the cake was a back-breaking sledge hammer operation.

Operation of the Huntingdon and Heberlein blast roasting pot was not without difficulties:

Fig. 6—Left, Smoke and fume were serious problems when the Heberlein pot was dumped.

as the product was too large and dense to be handled and channeling of air produced many pipes of overburned and underburned material. *C* illustrates a pressure-operated downdraft thin-bed rectangular furnace, which began to show the desired results. *D* is the same bed with suction used.

The suction fan was of the type represented by the Roots blower. Wear on impeller and casing was excessive, and the experimenters returned to the principle of the blast pot, using a thin bed as shown at *E*. Upward air currents through the bed created a fluidizing condition that prevented the highly desirable agglomeration through incipient fusion of adjacent particles as they momentarily came up to a sintering temperature. An effort to maintain quiescence in the bed by using an overlay of heavy screen proved successful. This generic sintering process, however, was not the Dwight-Lloyd process employed today because it was not continuous. Continuity was the goal.

For many weary days the inventors tried to combat the thin fluidized condition of the bed caused by the upflowing air. At last they concluded that the process must be used with a downdraft. Pallets were put in the charging hole after the hand screw had been turned, and the material was carefully spread on a shovelful at a time. A blow torch flame was directed on the bed through an opening, and the sinter, pallet and all, fell off the end.

Three people comprised the operating crew. In Fig. 9 the charger stands on top, the operator stands with his hand on the screw crank watching the timing for each operation, and the man at the left holds a small board used to smooth off the bed before the blow torch flame is applied. A wheelbarrow holds material for a shift run, and at right a box contains sinter from the previous shift.

At Cananea, Mexico, the first commercially operated machine, Fig. 10, started production in June 1906. Sintering on the lower strand, it had a hearth layer hopper, a feed hopper, an igniter, a suction box below the bed, and a hood over the bed.

This pioneer machine did the job but could not be fitted into the rapidly evolving high production straight line processing so characteristic of American practice. Straight line link and pin arrangements were tried, but the nature of materials handled and general corrosive and abrasive conditions resulted in high maintenance costs. To meet these difficulties the machine known as the Dwight-Lloyd sintering machine was designed.

Basically the machine consisted of a drive end with a speed regulator, feed hopper, igniter, and driving sprockets to push the individual ore-carrying pallets across the machine. Directly ahead of the drive end an active section was located. It comprised the suction, or windbox area where air was drawn down through the ignited layer, or, in some cases, was blown upwards. This section could be of any length required to complete the reactions and by adding additional windboxes capacity could be increased. The final section was the end where the individual pallets discharged the sinter and were guided to the lower track where they returned to the main sprockets by gravity. A gap to allow pallets to move rapidly into the curve was usually provided to assure the complete discharge of sinter.

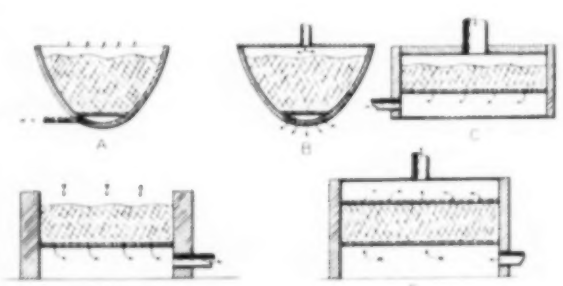

Fig. 8—Many experimental designs were tried by Dwight and Lloyd. Starting with the Huntingdon-Heberlein principle, the idea of a thin flat bed and a downdraft gradually evolved.

Fig. 9—The first continuous sintering machine in operation at Cananea. The operating crew consists of the charger, on top, the timer, with his hand on the crank, and the man at the left, who smooths the sinter mix with a board before a blow torch flame is applied.

This gap also allowed for expansion and contraction in use.

This design, with its individual pallets being pushed without any limiting connectors between moving members, provided an automatic take-up for wear and a means of removing any sections damaged in use.

Alleviation of dust, production of some sort of an agglomerate, and the controlled burning of volatiles such as sulfur was the total role of the old machines. But the impact of this system on metallurgical practices was important. Anything that a sintering machine would produce was so much better than what industry had been forced to use previously that design thinking became static. A long series of plants, each duplicating a former one, were built without much thought being given to the problems the individual plant had to face.

Little attention was given to controlled charge preparation and mixing. Rough volumetrically proportioned charges handled through such devices as a flat rotating plate with a few rabbling arms above it to turn the material over, gave way to the pugmill.

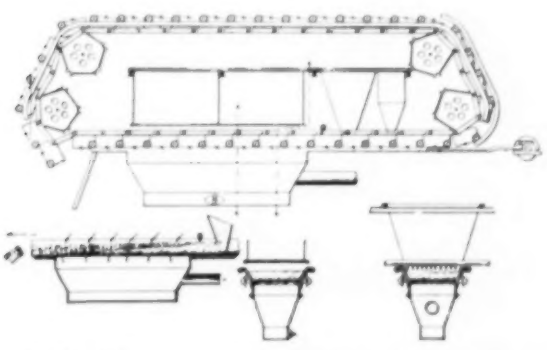

Fig. 10—The first commercially operated Dwight-Lloyd machine, June 1906.

Rotating drums with and without paddle shafts in them followed and began to show the value of controlled bed permeability produced by proper proportioning and mixing.

The value of preparing fine ores for iron blast furnaces and of agglomerating flue dusts was studied, and about four years after the introduction of the Dwight-Lloyd sintering system a ferrous plant was designed. A series of charge bins, each with its feeder, placed material on a collector belt. This discharged into a primary or dry mix pugmill followed by a secondary pugmill which had a roll crusher ahead of it to break up lumps coming from the primary pugmill. From here the mixed charge went to the sintering machine and then to a round cooling table which had rabbles on it so the sinter could be turned over for efficient contact with the water sprays.

The nonferrous industry quickly saw the value of controls and gradually incorporated various improvements. The ferrous industry continued sintering their flue dusts and eventually more of the fines which were contributing to the dusts. But extensive studies of the full problems were bypassed and the sinter plant served much as an incinerator handling waste materials.

Today, ferrous sintering is of such importance to efficient iron production that necessary studies have started. They are being undertaken in pilot-research laboratories such as those of the U. S. Steel Corp. in their new Monroeville, Pa., Research Center and the Dwight-Lloyd Research Laboratory of McDowell Co. in Cleveland. (See p. 843)

The original straight line Dwight-Lloyd sintering machine has proven so versatile that it has met the needs of calcining, firing discrete pellets by both up and down draft, and other heat treating processes of a similar nature. Today's sintering machines range from 12 in. to 12 ft in width and to over 200 ft in length, but the design principles remain the same.

Ore Preparation Plant of The Steel Co. of Wales Ltd.

by K. C. Sharp and H. R. Tufnail

THE ore preparation plant at the Margam ironworks section of the Steel Co. of Wales Ltd. commenced production in May 1954. The plant is designed to prepare all foreign ore and can be divided into three sections: Ore Crushing and Screening, Coke Breeze Preparation, and Sinter.

Ore Crushing and Screening Plant

The foreign ore prepared in this plant consists mainly of Swedish magnetites, North African hematites, and French siliceous hematites. All are free running except in really wet weather.

The plant consists of two parallel crushing lines and a purchased fines line, which bypasses both crushers and screens. The crushing lines are similar and, because they feed to the same belt, must handle the same ore when both crushers are working. The crushers are designed to reduce maximum size 10-in. ore to —2-in. in two stages; a primary gyratory cone crusher produces a —4-in. product and a similar secondary,—2-in. These crushers perform well with the hard ores for which they were designed.

K. C. SHARP and H. R. TUFNAIL are with the Steel Co. of Wales Ltd. at Port Talbot, South Wales. This paper was presented at the Blast Furnace, Coke Oven, and Raw Materials Conference, April 9-11, 1956, Cincinnati.

To lessen the load on the crushers, the ore is scalped to remove the natural fines before each stage of crushing.

In each crusher line, there are two primary and two secondary screens and the ore feed is normally split for each pair, but should there be any extensive mechanical repairs it is possible to divert the feed to one primary and one secondary only. The primary and secondary screens are independent of each other, the present settings being 1-in. screen on the primary and $\frac{3}{8}$-in. screen on the secondary. The oversize is transported to the blast furnace high-line bunkers and the minus $\frac{3}{8}$-in. fines to the sinter-plant stockyard. Each line is rated 500* tons per hr, but this depends upon the grade of ore being handled. At present, with the existing screens, the plant is handling between 250 and 300 tons per hr per line.

The purchased fines consist of material such as Sierra Leone and Sydvaranger concentrates and natural soft ore fines. The line contains three 100-ton receiving bunkers feeding onto a 1000-ton-per-hr transport belt, which bypasses both crushers and screens.

* It is assumed that all tons are long tons.

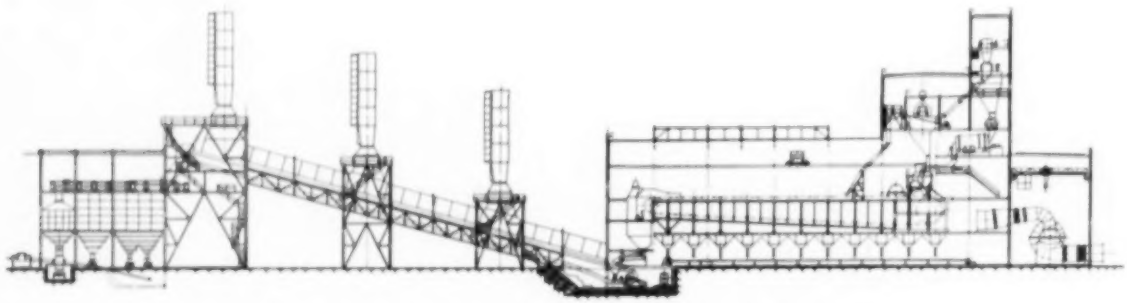

Fig. 1—The sinter plant comprises two strands, two Lurgi straight coolers, two fans, and a single stock.

Communications throughout the plant are provided by a common amplifier broadcast circuit with microphones and loudspeakers at all strategic points. Control of the plant is exercised from the crusher house, where the control panel is situated. Once the plant is started, no further control is exercised from this point. Adjacent to the control panel is a mimic diagram showing the layout of the machinery and belts, small lamps indicating whether the machinery is running or not.

Fines from the crushing and screening plant are transported by one of two belts to bunkers or into the stockyard. The yard is capable of holding 50,000 tons of fines and this, together with 12 ore bunkers, gives a total stocking capacity of about 62,000 tons.

As far as possible, the various groups of ore are kept separate at the sinter plant, where stocks are held of four main types. The concentrate stock consists of Sierra Leone together with Sydvaranger concentrates. All Swedish ores are grouped in one Swedish stock with the exception of Gellivare H, which is stocked together with other siliceous ores in a siliceous stock. North African hematites, together with purchased Benisaf fines, form the soft ore group.

Reclaiming from the stockyard is by grab gantry crane, rated at 250 tons per hr, with a grab load of 9 tons. The grab discharges into a hopper sited on the crane structure, and the fines are then carried to the bunkers by means of conveyor belts.

Fig. 2—Each sintering strand has 92 pallets and 19 windboxes.

Coke Crushing and Screening Plant

Minus 1¾-in. coke, dumped by either road or rail into the receiving hoppers, is first screened on a ½-in. primary screen. The oversize can be sent either to sale or to the primary crusher. The primary crusher is a toothed roll crusher designed to reduce this coke to −½-in. when it is recirculated to feed the primary screen.

The −½-in. coke is screened on a ⅛-in. electrically heated vibrating screen, to remove the natural fines. The purpose of electrically heating this screen is not to dry the coke but to ensure that the screen mesh stays clean. The oversize from this screen, −½-in. + ⅛-in., is fed into the two secondary roll crushers. These are made of smooth manganese steel and are adjustable to give an output of ⅛-in. This crushed coke joins the natural fines from the secondary screen and is transported by belt to the coke bunkers in the sinter plant. These bunkers hold a maximum of 240 tons.

In addition to coke, this section also handles flue dust and other admixture materials. These are discharged into separate receiving bunkers and fed to three 1000-ton admixture bunkers. The state of affairs where this section can handle either coke or flue dust at any one time is not satisfactory. Arrangements are being made to transfer the flue dust to the fine ore stockyard.

The general operation of this section of the ore preparation plant has not been up to the required standard. The main trouble points are the electrically heated secondary screen and the primary crusher. The secondary crushers need careful and frequent maintenance to produce reasonably sized material. It may be necessary to install rod mills, with which the operator is able to control and maintain quality even if at the expense of output.

Sinter Plant

The sinter plant comprises two strands, each with a grate area of 968 sq ft, two Lurgi straight coolers, two fans, and a single stack, Fig. 1. There are two gathering belts, one for ore fines and the other for flue dust, limestone, coke, and return fines.

The ore gathering belt is situated beneath the 12 ore bunkers from which ore is fed by circular feed tables powered by constant-speed motors. The feed tables are some distance above the gathering belt, and between the two is a chute which is movable to allow the check weigher to be driven from one bunker to the other. The check weigher is an adjustable belt weigh feeder mounted on a carriage; it is calibrated from 5 to 100 tons per hr and used to set the ore feed from each bunker twice per shift and whenever there is a variation in the total ore feed. It is moved under the discharge chute of the selected bunker and adjusted on the scale to the required ore tonnage per hr. The feed-table plough and skirting are then varied by the operator until the check weigher reaches its balance point.

Before the ore mixture passes into the primary mixer, a belt weigher indicates and records the total quantity of ore passing over the gathering belt. Dependent upon the ore make-up specified, there are a series of mixes, each with the same percentage of various ores but differing by 20 tons total wt per hr.

Dust and spillage from the wind main, dust legs, and cooler are stored in a 300-ton bunker as machine fines. These materials are too dusty to be weighed accurately; they are checked by switching off this particular table and noting the variation in the total belt weigher indication.

Feeding into the primary mixer from the opposite side is the admixture gathering belt. The first material fed on to this belt is the return fines from the 350-ton bunkers under the screens. The return fines are weighed by belt weigher, and the flue dust and limestone fines are fed on top of them. The latter two items are moved from 1000-ton bunkers by an ordinary feed table and cannot specifically be weighed. The last material put on this gathering belt is the coke. There are two belt weigh feeders to each coke bunker, and they are provided with a control panel which allows weight to be set to an accuracy of 56 lb per hour. These weighers are fitted with alarm devices to indicate an empty bunker or a faulty balance on the actual belt weigher. The combined coke and admixture feed passes over another belt weigher, which provides the method of checking the feed for flue dust and limestone by difference.

The ore and admixture feed join at the primary mixer, a 25-ft long trommel mixer without paddle but with blade.. fitted to the periphery of the drum to improve mixing. The mixer diameter varies from 10 to 15 ft and may be bypassed in case of trouble.

The mix is transported by belt to the top of the sinter building, where it is split by a reversing belt into two 80-ton surge hoppers. This reversing belt feeds for 4 min into each hopper. Before each surge hopper is a 1½-in. screen to remove any oversize material. Oversize is deposited on the sinter strand on top of the bed and is not sintered. Each hopper is lined with tiles, and inside is an open-ended box, directly under the screen feed, to prevent segregation within the bunker.

Exit from the surge hopper is by roll feeder, the speed of which can be finely varied, providing control of the feed into the secondary mixer. This is a trommel mixer with an off-center paddle at the entry end. The discharge end is plain, with the object of forming small pellets by the rolling action in the mixer. Angle iron has been welded along the length of the mixer to encourage the formation of a smooth layer of mix on the shell and provide a wearing surface.

During operation, the aim is to overfeed the surge hoppers with raw mix and stop the feed for short periods when the hoppers are full. This obviates frequent changes in rate of ore feed, and an ideal state of affairs would be one 10-min stop of feed per shift. In practice, the feed is stopped some 10 min per hr.

Final water additions are made in the secondary mixer. With the nonporous ores and concentrates normally used, optimum water content is 8 pct.

A swinging belt conveys material to the small surge hopper on the strand, distributing material across the hopper to prevent segregation. With a strand width of 8 ft 3 in., it is essential to have some form of antisegregation device, and each strand is being modified to include a paddle in the hopper. The normal bed depth on the strand is 12 in., made up of 1½ in. of ⅝-in. to 1-in. hearth layer, fed on to the bars before the mixture, which makes up the other 10½ in. Each strand, Fig. 2, has 92 pallets, 19 windboxes, and an ignition hood fired with blast furnace and 10 pct coke-oven gas. The operator controls the speed of the machine and the bed height to complete sintering on the 18th windbox and maintain the fan load at 1900 kw. Under these conditions the fan suction varies between 38 in. and 40 in. w. g.

Each strand has a fan driven by a 2650-hp motor rated at 245,000 cu ft of waste gas per min, at 40-in. w. g. suction and 150°C. The coupling between motor and fan is hydraulic and controlled from the strand operating desk.

The first and last four windboxes of the strand are dedusted by multiple cyclone dedusters, and the remaining windboxes have S-type dedusters. There is an expansion chamber in the wind main before the double entry to the fan, and the butterfly valve is between this expansion chamber and the fan. The exit from the wind legs is a manually controlled double flap. The upper shuts off the suction from the box, while the lower opens the exit. A cam system ensures that the upper flap is shut before the lower one can be opened, Fig. 3.

The strand pallets run on rubber rails with steel cladding, designed to take 90 pct of the load, leaving a ⅛-in. gap between the solid slide rails and the wearing pad of the pallet. This gap is filled with grease supplied by automatic pumps to provide the air seal of the strand. The grease consumption runs at 3.55 oz per ton sinter.

Fig. 3—The exit from the wind legs is a manually controlled double flap.

There is a toothed breaker at the discharge end of the strand. The sinter falls via impact boxes onto the feeder of the cooler. The cooling conveyor is a series of metal trays with mesh and louvered bottoms, which pass under a metal canopy containing three fans. These fans draw air up through the sinter to cool it. There is a certain amount of fines spillage through the cooler, because the sinter is not screened at the entry point. Such spillage is removed by a series of shaking trays and deposited into the machine fines system at the bottom of the cooler.

The speed variation control desks and instrument panels are housed back to back between strands, in line with the ignition hoods. At these desks, the operator makes the necessary changes of strand or

Table I. Sinter Mixes and Operating Data

Period of Time	Concentrates, Pct	North African Hematite, Pct	Swedish Magnetite, Pct	Siliceous Hematite, Pct	Return Fines, Pct	Fine Dust, Pct	Dolomite Pct	Coke Breeze, Pct	Sinter, Tons	Tons per Hr
Year ending Oct. 1, 1955	18.24	22.77	12.41	5.48	31.23	2.30	1.19	5.62	642,341	87.11
Week ending:										
Oct. 16, '54	25.65	15.51	11.82	5.38	33.59	2.11		5.94	11,350	76.78
Oct. 30, '54	24.28	14.76	10.99	7.91	33.72	2.36		5.98	11,163	80.99
July 16, '55	18.85	28.66	6.24	5.40	31.46	1.99	1.86	5.54	16,052	86.38
April 2, '55	16.87	29.01	10.73	2.53	30.73	2.83	1.29	6.01	12,323	87.45
Nov. 27, '54	15.19	16.81	21.84	6.76	30.53	1.77	1.12	5.98	11,473	90.16
Dec. 18, '54	13.51	19.86	17.37	8.36	29.60	2.15	3.31	5.84	13,226	96.45

cooler speeds, and the instruments to guide him are mounted on this panel. In addition, there is a mimic diagram and control desk, Fig. 4, housed in a pressurized cabin. From this desk the machinery is started and stopped.

The cooled sinter is discharged onto vibrating screens and split into four size fractions: return fines, $-\frac{3}{8}$-in.; blast furnace sinter, $\frac{3}{8}$-in. to $\frac{5}{8}$-in.; hearth layer, $\frac{5}{8}$-in. to 1-in.; and blast furnace sinter, +1-in.

These sizes are stored in bunkers, the return-fines bunkers catering for any variation in sinter quality during the shift. The hearth-layer bunker holds some 300 tons, and any surplus feeds into the sinter bunker. The sinter bunker is designed to hold 100 tons of sinter and thus allow intermittent feed to the furnace high line. The vibrating feeder from this bunker to the belt is a 1-in. punched-hole plate, which permits the fine sinter to fall on to the belt first and provide a bed for the warmer large pieces of sinter. The average temperature of the sinter on this belt is 100° to 150°C, but the center of some of the large pieces of sinter can be red hot. This temperature causes no difficulty from belt wear unless the belt is stopped full of sinter.

The ore mix with which the plant is worked is governed by the types of purchased fines available, the policy being to augment fines from crushing and screening with sufficient purchased fines to maintain the plant in production. Each strand has achieved 12,000 tons per week or better, and it is hoped that a plant output of 22,000 average with peaks of 24,000 tons and over will be achieved.

Table I shows the composition and variations in the mix over a 12 month period, together with relevant operating data.

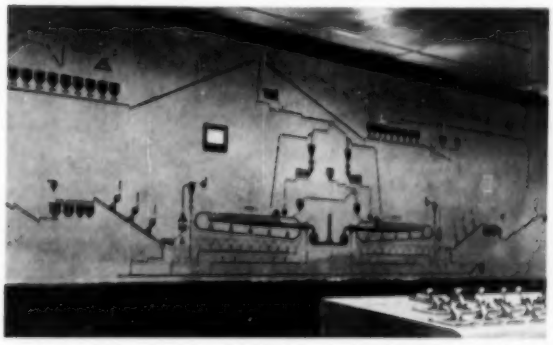

Fig. 4—The mimic diagram indicates immediately any breakdown in the sinter plant.

Effect on Blast Furnace Performance

Sinter has been in regular use in furnace burdens since May 1954, but the quantity available has never been sufficient to keep pace with demand. By January 1956, the plant had produced more than 1,000,000 tons, although this tonnage represented only 27 pct of the ore burden used during the same period. The Margam furnaces produce a high-grade basic iron and have burdens essentially similar to those of most other British furnaces; e.g., mixed foreign ore and home ore. The quantity of sinter is usually proportionally divided between the three furnaces.

Table II gives details of the average burdens used during January to April 1954, with no sinter, and June to December, with an average of 22 pct sinter.

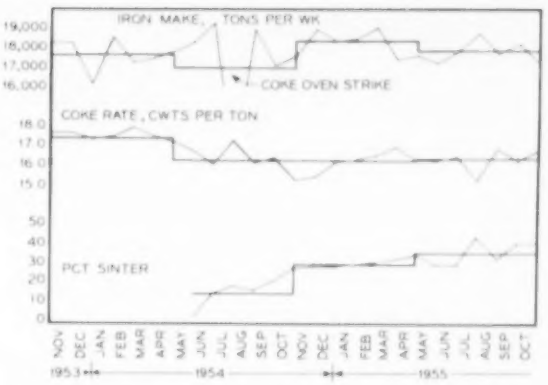

Fig. 5—Influence of sinter on the coke rate and iron make of Margam blast furnaces.

During May to October 1954, the percentage of sinter was gradually increased to around 30 pct, and no operating difficulty was experienced. Stock descent was regular, output continued at a high level, and the coke rate dropped by 1½ cwt per ton.

Fig. 5 shows monthly performance from November 1953 to October 1955 inclusive and indicates average weekly production, plant coke rate, and percentage of sinter. The six month averages are also shown. Throughout the whole period, the ores were similar, the average Fe content of the ore mixture being 46 to 48 pct and of the burdens 42 to 43 pct. The slag volume varied between 10.5 and 14 cwt per ton.

Tables III and IV give operating results for periods using varying percentages of sinter in the burden. In every case when the burden contained more than 50 pct sinter, extremely good coke rates were obtained. However, as expected with lower coke rates, stock descent was somewhat irregular, and as the sinter was increased, the necessity for checking became more apparent. With 70 pct sinter and over, regular checking every 15 to 20 min proved necessary for long periods. There were occasions when the blast pressure backed up beyond the limit of the blowers (75,000 cfm at 25 psi), and it was necessary to open the snort to prevent surging. Throughout the trial period, current blowing power was insufficient to determine the maximum blowing rate and hence output.

The effect of rate of drive is indicated by the Freyn rating on coke rate and performance. No. 1 furnace rate is restricted to 70 to 75 pct because of stove limitation while each stove is being rebuilt in turn. No. 2 and 3 furnaces normally operate at ratings around 90 pct. With only 50 pct sinter in the

Table II. Average Burdens, Margam Blast Furnaces, 1954

	January-April	June-December*
Iron make, avg per week, tons	17,810	18,369
Coke rate, cwt per ton	17.58	15.98
Carbon rate, cwt per ton	15.62	14.28
Burden, pct:		
Imported ores	65.48	44.14
Sinter		22.21
Home ores	29.01	28.21
Basic slag	4.70	3.92
Manganese	0.38	0.38
Scrap (Hecketts)	0.43	1.14

* Excluding July—strike at coke ovens.

burden, No. 1 furnace operated for many weeks with coke rates around 14.0 cwt per ton, whereas 75 to 80 pct sinter proved necessary to obtain similar performance on the larger furnaces.

This performance is considered close to if not optimum for current Margam practice. Further improvement is unlikely while some 25 pct of the burden is unprepared home ore (22 pct Fe) together with carbonate fluxes in the burden. Beneficiation of the home ore is not yet practiced nor even planned, although pilot-scale trials are being carried out.

Fine limestone formed part of the sinter mix during Dec. 1955, and future burdens will contain an increasing amount of self-fluxing sinter.

The physical condition of the sinter measured at the furnace skips proved a reliable guide to operations, and dusty sinter was soon followed by periods of stiffness. Fig. 6 shows histograms of sinter size during 1955 measured at the sinter plant and furnace skips. The amount of size degradation in transit has not yet been solved but is receiving close attention. During late 1955, sinter was screened at the furnace high line in order to improve the size range and thus affect better performance. It is suggested that future practice follow this lead. Plants using wholly sintered burdens may dispense with the conventional high line, scale cars, and feed both sinter and coke via screens and weigh hoppers to the charging skips.

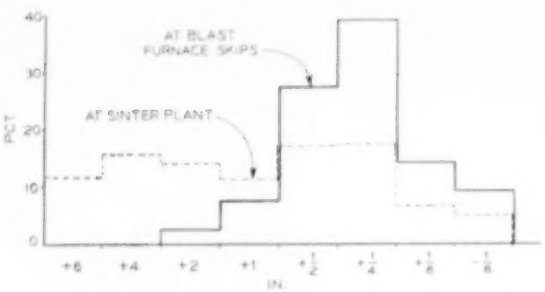

Fig. 6—Sinter size analysis.

Iron Quality

Low-silicon, low-sulfur iron has always been the target at Margam, and as the melting shops have only inactive mixers, there is every advantage in reducing their metallurgical load. As many ores in current use are medium to high in phosphorus, it is extremely difficult to produce a low-phosphorus iron, 0.4 to 0.5 pct being among the lowest produced, while the average is 0.7 to 0.8 pct.

Fig. 7 and Table III indicate the increase in the percentage of casts below 0.50 pct Si and the percentage below 0.03 pct S during the period sinter was introduced into the burden. For most of 1955, the average daily sulfur rarely exceeded 0.025 pct, partly because of slags of high desulfurizing potential. The very low sulfur contents obtained from sinter burdens follow the reduced coke rate and thereby lower sulfur input. It had been suggested that the regular checking necessary with high sinter burdens would cause some falling off in iron analyses. However, the grade has been maintained and improved following the greater use of sinter.

Elliot[1] has suggested that sinter be regarded as a premelted slag and has drawn attention to the necessity of control over the sinter mix to produce the most desirable sinter composition for a given iron making practice. With high sinter burdens, the normal control of the process has largely passed from the blast furnace operator to sinter-plant personnel. Due attention to the diagrams[2] enables a sinter composition to be produced, which, on melting, leads to a slag of low melting point. Changes in sinter composition cause the resulting slag composition to vary and should be avoided. As has been experienced elsewhere, during periods of steady sinter analysis, furnace operation has greatly benefited and troubles largely disappeared.

Table III. Comparison of Practices, Twin Furnaces Nos. 2 and 3, Hearth Diam. 25-Ft 9-In.

Furnace	No. 2	No. 3	No. 2	No. 3	No. 2	No. 3
	May 1953	June 1953	Jan 1954	June 1954	Nov 1954	July 1955
Number of weeks	4	5	4	5	4	4
Tons per day	968	881	966	1,006	1,017	985
Tons per day per sq ft hearth area	1.859	1.692	1.855	1.932	1.953	1.891
Freyn rating, pct	95	95	85	85	80	80
Coke rate, cwt per ton	19.19	20.81	16.72	16.19	15.05	14.76
Slag volume, cwt per ton	13.78	13.30	12.02	11.80	9.59	11.17
Burden, pct:						
Imported ores	54.66	62.86	65.43	51.33	31.33	11.83
Home ores	40.54	31.70	29.52	28.84	24.06	18.77
Sinter				14.12	42.89	63.19
Basic slag	4.58	5.20	4.76	4.65		5.15
Manganese	0.20	0.24	0.29	0.28	0.52	
Scrap (Hecketts)				0.78	1.20	1.06
Ores, cwt per ton iron	52.15	49.27	46.25	43.55	40.50	39.30
Fluxes, cwt per ton iron	1.02	1.65	2.40	2.41	3.85	4.26
Percentage Fe, ores only	38.35	40.59	43.24	45.93	49.38	49.34
Percentage Fe, total burden	37.61	39.28	41.11	43.52	45.09	44.48
Average blast volume, cfm	74,944	74,541	68,137	72,428	64,402	65,746
Average blast pressure, psi	17.00	15.6	15.30	15.80	15.45	22.20
Average blast temp., deg F	1,027	1,107	1,116	986	1,042	1,137
Flue dust:						
Lb per ton iron	397.48	279.46	156.63	192.23	88.05	175.59
Blast off	11 hr 19 min	14 hr 33 min	15 hr 09 min	6 hr 49 min	9 hr 08 min	36 hr 20 min
Availability, pct blowing time	98.32	98.27	97.75	99.19	98.64	94.59
Sulphur input, lb per ton iron	24.43	26.25	24.58	22.95	21.28	19.09
Average iron analysis:						
Si	0.56	0.76	0.52	0.40	0.42	0.37
S	0.051	0.064	0.038	0.035	0.033	0.019
P	0.89	0.87	0.65	0.70	0.44	0.47
Mn	1.11	1.08	1.09	1.25	0.91	1.40
C	4.07	4.04	4.06	4.03	4.11	4.09
Pct casts 0.50 Si and below	35.77	9.03	47.54	81.01	70.08	81.70
Pct casts 0.03 S and below	3.25	4.52	27.87	36.71	56.69	92.81

Table IV. Comparative Performance Data, High Sinter Burdens, Margam Blast Furnaces

Furnace	No. 1 (21' 6")	No. 2 (25' 9")	No. 3 (25' 9")
Week ending	Sept. 24, '55	Aug. 13, '55	July 16, '55
Iron make, tons	5,089	7,050	8,046
Tons per day per sq ft hearth area	2.006	1.934	2.207
Freyn rating, pct	70	85	84
Coke rate	14.04	14.54	14.02
Blast volume:			
Maximum	49,500	74,000	67,694
Minimum	43,500	65,000	60,000
Average	46,314	69,548	65,185
Blast pressure:			
Maximum	20.0	23.5	26.0
Minimum	16.0	17.7	22.0
Average	17.1	19.5	23.0
Average Si in iron	0.40	0.37	0.31
Casts below 0.05, pct Si	77.14	94.59	90.48
Average S in iron	0.019	0.019	0.027
Casts below 0.03, pct S	100.00	100.00	100.00
Slag volume, cwt per ton	11.69	10.60	10.56
Fe in ores, pct	48.12	54.83	50.72
Sinter, pct	46.15	77.75	71.13
Blowing time	167 hr 30 min	150 hr 32 min	168 hr 00 min

Blast Temperature

The furnace operators at Margam always dealt with a hanging or stiff furnace by reducing the blast temperature, and experience over many years supported the practice. The resulting upward extension of higher temperature usually melted any material on bosh wall or lower stack. Both Elliot[1] and Danielsson,[3] operating with sintered burdens on widely different practices, have indicated that high blast temperatures of the order of 750°C to 850°C are possible, thus effecting maximum economy.

With high sinter burdens, the blast temperature also increased at Margam, although regular temperatures over 650°C have not yet proved possible. This is perhaps due to the older type of foreman, who still favors former practice. But most of the operators now recognize that high ore-coke ratios, and the resulting coke rates, call for high blast heat and regular checking when necessary.

Blast Pressure

Reference has been made to the higher blast pressure accompanying increased percentage of sinter, and there have been occasions when the pressure limit has been reached. On such occasions, the stock column tightened to such an extent that less air was being blown, and therefore output was restricted, owing to the lower quantity of coke consumed at the tuyeres. In every case, this could be attributed to periods of poorly sized sinter. A slight alteration in burden soon restored stock movement. The operators are convinced that blowers should be capable of maintaining the desired volume at much higher pressures in order to force the blast through the burden even during such periods of stiffness.

Prior to using burdens containing sinter, if the blast pressure was increased above 18 psi, the stock column tightened and either blast temperature or volume, or both, were reduced in order to promote and restore free driving. With 50 pct sinter or more, the pressures often rise over 22 to 23 psi before a similar condition is experienced. Often the pressures remain around 24 psi, and the furnace is checked every 20 min to promote stock descent. The increased blast pressure most probably follows the higher ore-coke ratios permissible from high sinter usage. The reduced coke consumption inevitably means less coke present in the stack, thereby slightly reducing voidage. High blast pressures are therefore more likely with a low Fe content burden (higher ores per ton) than with a high Fe content burden. During trials with high sinter burdens it was decided to operate for a few weeks with richer burdens (from 48 pct to 55 pct Fe). A decided improvement in stock descent was noted, and blast pressures were some 2½ lb lower.

Further knowledge is necessary on the subject of blast pressure but will be possible only through experience with blowing at much higher pressures. This will be possible with the new large furnace.

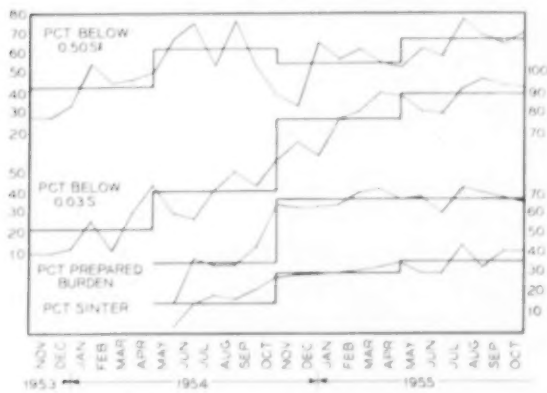
PCT BELOW 0.505i
PCT BELOW 0.035
PCT PREPARED BURDEN
PCT SINTER
NOV DEC JAN FEB MAR APR MAY JUN JUL AUG SEP OCT NOV DEC JAN FEB MAR APR MAY JUN JUL AUG SEP OCT
1953
1954
1955

Fig. 7—Influence of sinter on sulfur and silicon of Margam pig iron.

Conclusion

The foregoing brief notes illustrate the making and effect of sinter on Margam practice. Because of the relatively short period and limitations of both sinter output and furnace blower capacity, further studies and operating experience are necessary. As at Appleby-Frodingham, future practice is expected to show increasing use of wholly sintered or beneficiated burdens for both lean and rich ore working.

The experience of English plants using low-grade ores and Swedish plants using rich ores, both in a wholly sintered and self-fluxing condition, have achieved enviably low coke rates, and are a distinct pointer to future practice.

References

[1] G. D. Elliot, J. A. Bond, and T. A. Mitchell: *Iron Making from High Sinter Burdens*. Symposium on Sinter. Iron and Steel Inst. *Special Report* No. 53.

[2] E. F. Osborn, R. C. DeVries, K. H. Gee, and H. M. Kraner: AIME *Blast Furnace, Coke Oven, and Raw Material Process*, 1953, vol. 12, pp. 281-317.

[3] C. Danielsson: *Sintering Practice at Domnarfvet, Sweden*. Symposium on Sinter. Iron and Steel Inst. *Special Report* No. 53.

J&L's Star Lake Sinter Strand

by R. G. Fleck and F. M. Hamilton

AS the result of extensive small laboratory experiments, the metallurgical results of sintering iron ore from the Benson mines at Star Lake, N. Y. were clearly defined. But the practical operation had not been proved; so, the sintering strand, which went into operation at the mines in 1952, was designed to minimize the risk involved. It was built by the Dwight-Lloyd Div. of McDowell Co. Inc.

Sintering Machine

The hot section of the sintering strand, consisting of 16 windboxes, is 112-ft long, while the cooling section, which includes 5 windboxes, is 35-ft long, and the machine has a width of about 6½ ft.

The hot section of the strand is serviced by a No. 23 Double Inlet American Blower fan, 115 in. in diam which is driven by a 1000-hp motor equipped with a hydraulic coupling to the fan. The rated flow of gas per sq ft of hearth area is 229 cfm.

The cooling section of the strand is serviced by a No. 19 Double Inlet American Blower fan, 91 in. in diam. It is driven by a 500-hp motor equipped with a hydraulic coupling. Rated flow of gas per sq ft of hearth area is 375 cfm.

Dust collection in both the hot and cooling sections is by means of American Blower No. 51 cyclones. Four cyclones, arranged in parallel, service the hot section, while two cyclones service the cooling section. The drive motors of each fan are cooled and air cleaned by a 380-watt Westinghouse Precipitron.

R. G. FLECK and F. M. HAMILTON are with New York Ore Div., Jones & Laughlin Steel Corp., Star Lake, N. Y. This paper was presented at the Blast Furnace, Coke Oven, and Raw Materials Conference in Cincinnati, April 1956.

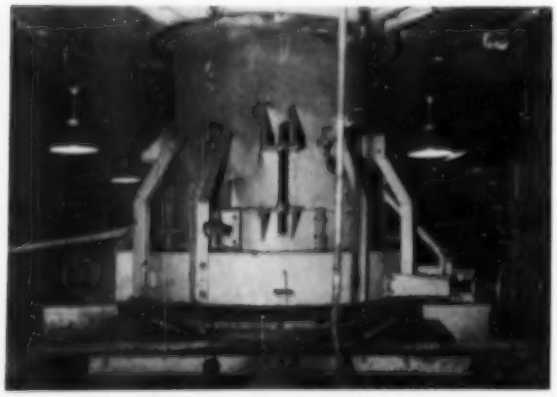

Fig. 1—Double discharge 72-in. table feeder for iron concentrates.

Raw Materials

The plant usually treats what are called martite concentrates, although it has operated at times on finer magnetite concentrates. Martite is the product of a spiral concentration process and originally came from the portion of the crude ore containing too much nonmagnetic material to make treatment in the magnetic concentrator practical. Typical screen and chemical analyses of the concentrates are given in Table I.

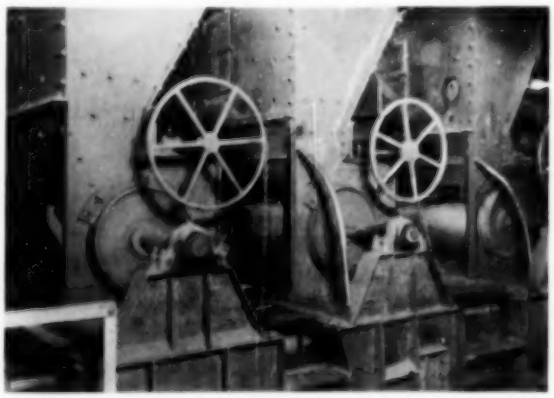

Fig. 2—Roll feeders for metering sinter mixes to the strand bed.

Fuel: No. 5 buckwheat anthracite culm is used for fuel in the mix, and No. 2 fuel oil is used for ignition.

Handling of Raw Materials: The raw materials are stored in four reinforced concrete silos. Three silos of 785 tons capacity each are used for concentrates and one silo of 185 tons capacity for anthracite culm. Each concentrate silo is equipped with a double-discharge 72-in. table feeder, Fig. 1, and the coal silo is equipped with two 24-in. wide belt feeders.

Mix ingredients are transported by two parallel 24-in. conveyors, leading to the pugmills located above the strand floor. Returns are added to each belt between the silos and pugmills. The mix is delivered to the strand in two separate streams, with fuel and water contents adjusted as required. The mixes are metered to the bed by roll feeders, Fig. 2. The third roll feeder is also available to feed a hearth layer to the strand.

The plant was designed to screen and crush returns to provide sized material for use as a hearth layer and as a mix ingredient for improving bed permeability. Returns from the grizzly at the discharge end of the strand are conveyed to a 4-ft by 10-ft vibrating screen. Undersize material (−⅝-in.) from the screen is sent to the strand. Oversize material is crushed in a 36-in. cone crusher with a ⅝-in. setting, and the crushed returns are combined with the original undersize product. The entire load of returns is conveyed to another screening station ahead of the strand. At this station, the returns are screened to ⅜-in., the oversize going to the strand as a hearth layer and the undersize being added to the mixes before pugmill treatment.

Handling Product

The product of the strand is partially air-cooled sinter, which requires only a little water to prevent damage to the paint of the railroad cars. When discharged, the sinter cake falls on a grizzly having 1-in. spacings and then to railroad cars for shipment to steel plants at Pittsburgh, Aliquippa, and Cleveland. Fig. 3 shows typical sinter from this operation, while Fig. 4 shows water-quenched sinter from the other strands.

Table I. Typical Analyses of Martite Concentrates

Chemical Analysis, Pct	
Iron (soluble)	60.52
Silica	6.14
Phosphorus	0.165
Sulfur	0.270
Titania	1.93

Screen Analysis		
Mesh	Pct Wt	Cum. Pct Wt
+ 14	0.4	0.4
+ 20	4.5	4.9
+ 28	14.9	19.8
+ 35	19.4	39.2
+ 48	18.9	58.1
+ 65	17.7	75.8
+ 100	12.2	78.0
−100	12.0	100.0

Discussion of Results

Air Cooling: At the time this plant was designed, research had shown the desirability of producing an air-cooled sinter. After investigation of the various possibilities, it was decided to extend the strand length beyond the original planned length and use this portion of the strand for air cooling. Separate wind systems were provided for each section of the combined strand, even though the transition from the burning to the cooling area merely involved movement of the pallet to an adjoining windbox. Thus it was felt that if the air-cooled sinter did not seem worth while to the consumers, the air cooling section of the strand could be used for burning and automatically provide increased sintering capacity.

The air-cooled sinter is highly satisfactory, and today it is a preferred product. In view, however, of more recent developments in air cooling equipment, strand cooling might not be the most efficient method to use. For future installations, consideration will be given to other types of cooling practices.

Sized Returns: The plant was designed to provide sized returns for the system, large returns for a hearth layer and smaller returns to promote bed permeability. It was found that air-cooled sinter made in this fashion would not normally provide enough returns, because of its greater resistance to breakage and abrasion. Wear of screens and crushers proved to be considerable. Therefore, while the use of a hearth layer of large returns and smaller returns in the mix is still felt to be attractive, the benefits obtained do not justify the cost of maintenance. For these reasons, the sized return system is not now used at Benson mines.

Fig. 3—Air-cooled sinter is highly satisfactory, and today it is a preferred product.

Multiple-mix Layers: The two-layer charge system makes it possible to take some advantage of two well-known facts: water vapor condenses in the lower and cooler layers of the mix, and the oncoming waste heat from above provides part of the necessary heat for fusion. Theoretically, a mix in which the fuel and water could be arranged in the proper gradation from top to bottom, between some known limits would be best for sintering. Lacking a practical method of attaining such a condition, this strand was provided with means for metering two separate and distinct mixes, one on top of the other. The bottom layer is run lean in moisture and fuel. The result is a smoother operation, some savings in fuel, and a more uniform product.

A by-product of the two-layer system is some saving of downtime. Pugmills require considerable maintenance, and while one pugmill is off for repairs it is possible to run the strand on mix supplied entirely by the other pugmill and metering system.

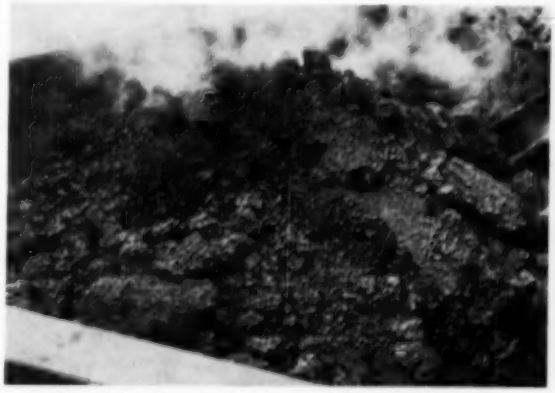

Fig. 4—Water-quenched sinter is still commonly used.

Weirton's New Sintering Plant

The first U. S. 8-ft sintering strand uses pelletized mix delivered by a vibrating slotted feeder.

by J. H. Strassburger

WEIRTON STEEL CO., Div. of National Steel Corp., started operation of the first 8-ft wide iron ore sintering plant in the U.S. in November 1955. This sintering plant is the continuous Koppers Aorco type, designed for the sintering of ore fines, flue dust, and miscellaneous materials.

The plant has a capacity of approximately 2500 tons of sinter per day, based on 3 tons per sq ft per day of active grate area. It has incorporated several special features, made necessary by a site restricted in size. The flow of materials through the plant, starting at the blast furnace highline, is shown in Fig. 1.

Hopper and Conveying System

Iron ore from the ore yard is delivered by transfer car to four track hoppers having a total capacity of 500 tons. Each of these four track hoppers is equipped with a 36-in. by 72-in. vibrating feeder. The ore is controlled to the feeder by a rack and pinion gate. The feeders are lined with stainless-steel to facilitate movement of sticky ores.

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The run-of-mine iron ore from the track hoppers is delivered to the screening station by three 30-in. belt conveyors arranged in series to handle 400 tons per hr. The No. 1 conveyor takes the ore from the track hoppers. It can go directly onto No. 2 belt or onto a scalping screening conveyor. Minus 2-in. ore is delivered directly to No. 2 belt through a chute, and plus 2-in. goes over the scalping conveyor to a 4¼-ft Symons cone standard crusher. The crushed ore is then delivered from the crusher back onto No. 2 belt. It is mixed with the screenings and delivered to No. 3 belt. This takes it to an overhead triple chute system, where the ore is transferred to the three ore screens.

Ore Screening Station

The iron ore screening station consists of three Tyler screens, two of the single deck rod type and the third a double-deck screen with a stainless-steel, heated bottom deck. The screening of the ores has been quite satisfactory, with no evidence of screen blinding when screening to the —⅜-in. size.

The rod deck screens consist of ⅜-in. diam, 36-in. rods, mounted in rubber holders with rubber supports. There are four sections of rods on the screen,

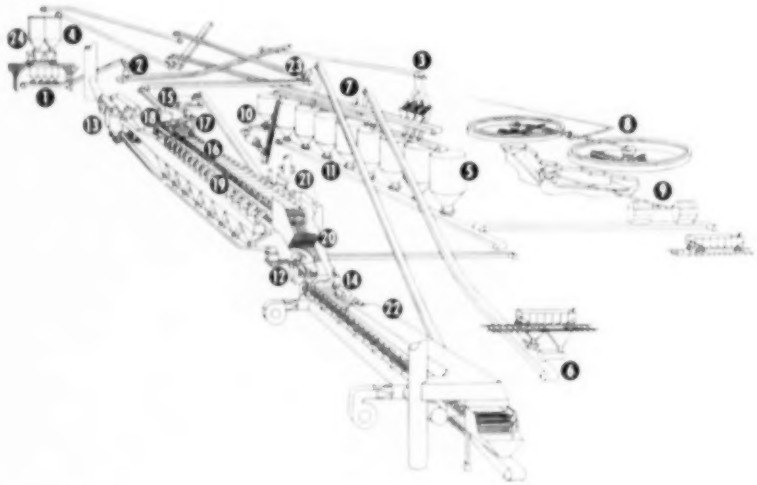


Fig. 1—The Weirton iron ore sintering plant comprises: 1) Highline ore track hoppers, 2) Ore crusher, 3) Ore screens, 4) Lump-ore storage bin, 5) Stock bins, 6) Miscellaneous materials, track hopper, 7) Rubble screen, 8) Thickeners, 9) Filters, 10) Rod mill, 11) Table feeders, 12) Primary pugmills, 13) Dust collectors and fan, 14) Mix conveyor, 15) Secondary pugmills, 16) Sintering machine, 17) Vibrating feeder, 18) Hearth-layer feeder bin, 19) Windboxes, 20) Sinter breaker and grizzlies, 21) Discharge end dust-collecting system, 22) Sinter cooler, 23) Sinter screens, and 24) Sinter storage bin. Not shown: grizzly screen at end of cooler for separation of open hearth sinter.

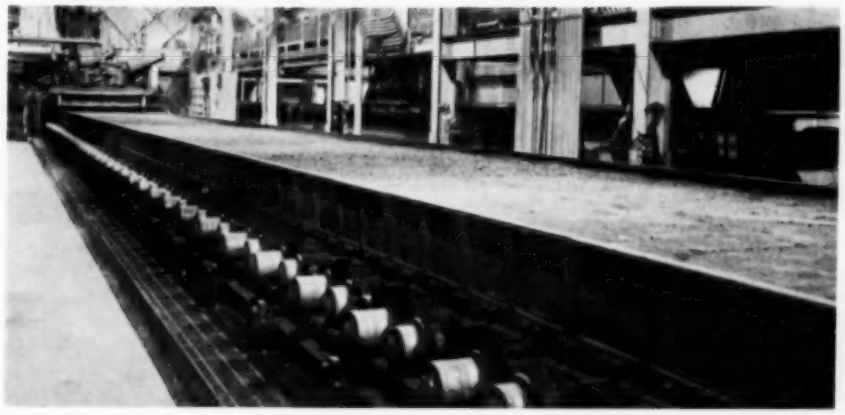


Fig. 2—The sintering machine is the continuous Koppers Aorco type, 8-ft by 104-ft, with 13 windboxes.

and the actual slotted opening is $\frac{3}{8}$-in. by 12-in. Rubber mounting apparently gives a secondary vibration to the rod, which prevents blinding with wet, sticky ores. Ore screenings are transferred by shuttle conveyor into the ore stock bins. Oversize ore may be delivered to an overhead ore-storage bin above the highline or directly to the ore yard. The highline bin feeds oversize to a transfer car, which delivers it back to the furnace bins.

Miscellaneous Materials

Materials such as flue dust, coke breeze, roll scale, or limestone fines are handled by a track hopper at yard level. The track hopper has two bins; one bin handles flue dust, and the other hopper handles the miscellaneous materials. A conveyor system carries these materials to an overhead screening station where the flue dust, coke breeze, and other materials are screened over a double-deck 6-ft by 12-ft screen. If desired, the miscellaneous materials can be bypassed around the screens for direct delivery to the stock bins. Screened material is delivered by a shuttle conveyor to a designated bin.

Coke breeze is taken from the bin by conveyor to a 6-ft by 12-ft Hardinge rod mill, where it is ground to $-\frac{1}{8}$-in. size at the rate of about 10 tons per hr. It is then delivered by skip hoist to the coke-dust storage bin.

Storage Bins

In addition to the coke-breeze bin, there are eight bins for storing ore fines, flue dust, coke dust, roll scale, and limestone fines. These bins have a capacity of about 500 tons based on iron ore. They are built with hoppers sloping 70° at the bottom to prevent difficulty with sticky, wet ore. Each stock bin is equipped with an 8-ft 6-in. diam table feeder which can be set by means of a plow. A variable voltage dc motor controls raw material feed onto the sinter-mix belt. Pan weights of all materials fed onto the belt system are taken every 30 min.

Pugmill System

Normally the sinter mix consists of approximately: 56 pct Labrador fines, 15 pct flue dust, 15 pct hot returns, 10 pct cold returns, and 4 pct coke breeze. The carbon in the mix is held at about 5 pct.

Because Labrador ore is limonitic, with 10 pct surface moisture and 7 pct ignition loss, the theoretical yield of sinter is about 83 pct. The sinter analysis averages about 64 pct Fe and 6 pct Si.

Sinter mix from the stock bins is fed by conveyors to the primary pugmills. These conveyors also receive hot returns, dust from the windbox hoppers and cyclones, and filter cake from the flue-dust recovery system. There are two pugmills, one operating and one spare. These mills are 12-ft 4-in. long, of double-shaft trough type with 26-in. diam Ni-Hard blades at 22-in. centers. Facilities are available for adding water to the primary pugmills with electric solenoid controls tied into the sinter machine circuit. Pugged material from the primary mills is delivered by conveyor to the secondary pugmills, which consist of a 6-ft diam by 13-ft 6-in. long balling drum as well as a trough pugmill that duplicates the primary mills.

A balling drum is utilized for preparing the sinter mix. It is driven by a dc variable motor drive and is lined with expanded metal for the formation of an ore coating against the shell. This mill originally was equipped with rotating paddles but these were removed in order to obtain better pelletizing. There is also a scraper bar for maintaining the proper thickness of ore layer against the shell. Moisture, controlled by water flow from a constant-head tank, ranges between 11 pct and 13 pct. The sinter mix, after leaving the secondary balling drum, is delivered by a conveyor to the sinter machine feeder.

Sinter Machine Feeder

From a distributing hopper the pugged mix passes onto a vibrating slotted feeder with an 8-ft long slot at right angles to the sinter machine. The feeder has a stainless-steel liner and is heated electrically. It delivers the feed onto the sinter machine grates against a striking plate, which is used to control the height of the bed, normally 14 in., including a 1-in. hearth layer.

The hearth layer, consisting of $\frac{3}{4}$-in. by $\frac{3}{8}$-in. sinter, is screened out of production and delivered by chute into a spreader hopper the width of the sintering machine. It is fed by gravity, its height controlled by an adjustable cutoff bar above the sinter grates. The use of a hearth layer is considered conducive to good grate-bar life; it also has the advantage of getting better air distribution, minimizing fines, and preventing the sinter cake from sticking to the grate bars when being discharged. The use of a hearth layer at Weirton has been very satisfactory.

Sinter Machine

The ignition furnace of the sintering machine consists of a row of torch burners using coke-oven gas with a pressure air blower. The burners are set in a refractory-lined water-cooled box.

The sinter machine is an Aorco continuous type 8-ft by 104-ft long, with 13 windboxes, each 8-ft long. The pallets are 32-in. long, of cast-steel construction throughout with a single row of finger-type Mallix grate bars, each 32-in. long. The machine is driven by a variable dc motor drive with a speed range of 4 to 12 fpm. Fig. 2 is a view of the sintering machine in operation, looking toward the charge end, with the ignition furnace and the vibrating feeder shown in the background.

Fan Exhaust System

The sinter machine suction system consists of a large rectangular duct connecting into the 13 windboxes. The last four windboxes have butterfly damper controls. The exhaust duct is connected into four cyclone dust collectors set in pairs. These cyclones are 10-ft diam by 15-ft and are completely lined with Carbofrax brick. The cyclones are equipped with electrically driven air-lock dust valves, which discharge the dust into vibrating tubular feeders to the dust-collection belt. The main exhaust fan is a double inlet type with motor-operated inlet louvers for control. This fan is rated 210,000 cfm at 30 in. suction and 280,000 cfm at 25 in. suction. It is driven by a 2000-hp motor at 690 rpm.

The discharge end of the sinter machine is completely enclosed in a steel-plate hood, which is connected to a Multiclone with a 20,000-cfm exhaust fan.

Fig. 3—The sinter breaker consists of a 10-in. diam shaft with rotating cutting blades and rakes for producing —8-in. sinter.

Discharge and Breaker

Sinter cake discharge from the sinter machine passes over two groups of grizzly bars set at ¾-in., three sets of grizzly bars being ahead of the sinter breaker and two sets following the sinter breaker. The sinter breaker, Fig. 3, consists of a 10-in. diam motor-driven shaft with rotating cutting blades set at 16-in. centers and rotating rakes between the cutting bars, so that actual spacing is 8 in. for producing —8-in. sinter. Sinter is discharged from the breaker over the second set of grizzly bars down a discharge chute into the feed hopper of the sinter cooler. Hot returns from the grizzly bar system drop into a brick-lined hopper, from which they are returned to the sinter mix belt system by a vibrating feeder.

Cooling and Screening

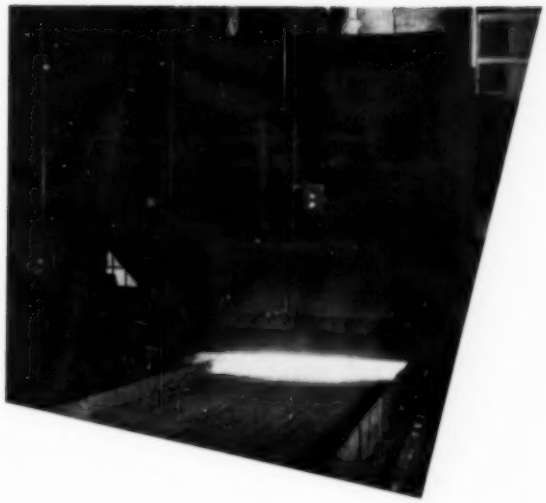
The sinter cooler is the forced-draft, chain-grate type, 6-ft wide and 150-ft long. The cooler is basically similar to a chain-grate stoker, so that individual links can expand and contract without danger of warping or breaking. The entire sinter cooler other than the hood, fan ducts, and stack system, is of cast-steel construction. It is driven by a variable dc motor with four-to-one speed range from 5 to 20 fpm. Sinter is cooled by a pressure air system delivered up through the grate bars by three fans, each rated at 85,000 cfm at 8-in. pressure. The sinter cooler, Fig. 4, is designed so that approximately 100,000 cfm of air is available per ton of sinter handled. The cooling effect has been satisfactory for delivery onto the belt system. Siftings are collected in hoppers under the cooler and are discharged to the sinter product conveyor system.

Fig. 4—The sinter cooler is the forced-draft, chain-grate type, 6-ft by 150-ft.

After leaving the cooler, sinter can either go by conveyor over a 2-in. spaced bar grizzly into railroad cars for blast furnace or open hearth furnace use, or to a sinter screening system, where —¼-in. cold returns are screened out as well as the ¾-in. by ⅜-in. hearth-layer material. The screen for cold returns consists of a 5-ft by 10-ft double-deck vibrating screen. The hearth-layer screen is also a double-deck vibrating unit. The hearth layer is obtained between the two decks, while the oversize and undersize materials go into production. After leaving the screening station, oversize sinter is delivered by belt conveyor to a 500-ton storage bin above the highline. This bin charges the transfer car for delivery of sinter to the blast furnace stock bins. All belt conveyors handling run-of-mine ore, oversize ore, and sinter production are equipped with weightometers for obtaining production data.

Flue-Dust Collection System

At the same time when the sintering plant was being laid down, Weirton Steel Co. also installed a complete gas-washer, water-collection system with pumps, pipe lines, thickeners, filters, and other auxiliary equipment for recovery of flue dust from the gas washer and precipitator waste waters. Each blast furnace is equipped with a 5000-gpm pump, which picks the water up from the gas-washer discharge sump and delivers, by a separate 18-in. pipe line, to a distributor box located between two 80-ft Dorr thickeners. The thickeners have an effective depth of 23-ft at the periphery and are rated to clarify the gas-washer water to a dust content of 2.5 grains per gallon. The sludge from the bottom of each thickener is delivered by one of two diaphragm pumps to a circulating pipe line system from which sludge can be bled into one of two Eimco disk filters. Each of the filters has seven disks 8-ft 10-in. in diam. The filter cake is discharged onto a shuttle belt conveyor, which can either deliver the cake to the sinter-mix conveyor ahead of the primary pugmills or to railroad cars.

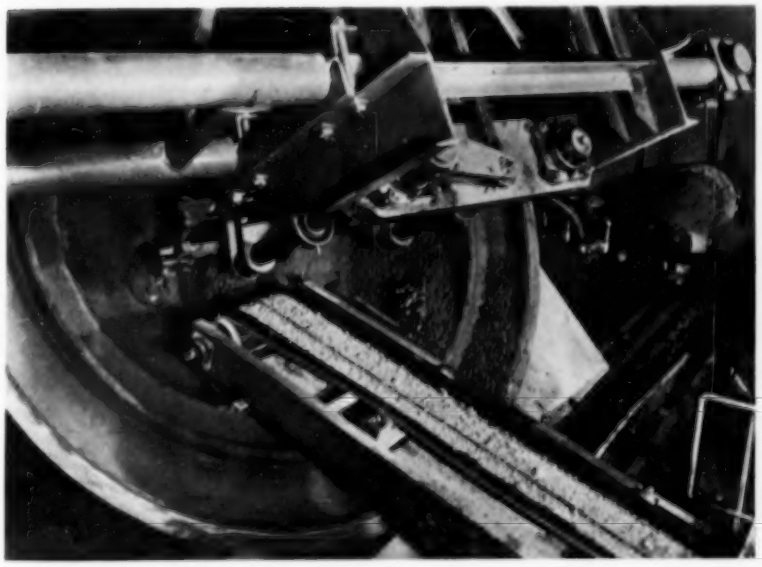


From Established Sintering Techniques

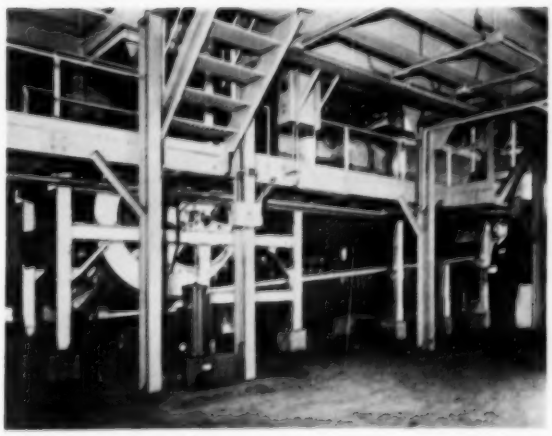
. . . a new laboratory has been designed to perfect and expand the scope of the sintering process.

The Dwight-Lloyd Div. of McDowell Co. has put into operation a complete mineral agglomeration laboratory and pilot plant with equipment for crushing, sizing and screening, pelletizing, heat-hardening pellets, sintering, blending of materials, strength analysis, and drying and calcining.

Currently under development is a disk agglomerator, or *flying saucer*, for agglomerating fines into a uniformly sized pellet for use in sintering. As the flying saucer revolves, the fines fed into the innermost saucer agglomerate. With increasing particle size, the pellets tumble over successive lips to the outer ring where they pass over the last lip and on to further processing.



Also under development is equipment to use the up-draft principle of sintering. The sintering strand pictured at right is equipped for such experimentation.



Sintering Iron Ores and Concentrates At Extaca

R. L. Bennett,
R. E. Hagen,
M. V. Mielke

THE Extaca plant, near Virginia, Minn., agglomerates fine iron ores and taconite concentrates by nodulizing or sintering. A previous paper[1] dealt with operation of the nodulizing plant. The present report deals with operation of the sintering plant, both on fine iron ore and on taconite concentrates.

Raw Materials

The process raw materials consist of iron ore fines, magnetite taconite concentrates, coke breeze, anthracite fines or other solid fuels, limestone, hydrated lime, and No. 2 fuel oil. Typical sieve and chemical analyses of these materials are given in Table I.

Practically all of the iron in taconite concentrates is present as magnetite. Quartz, iron silicates and carbonates, and some form of lime comprise the principal gangue constituents. Concentrates are relatively uniform in size and chemical analysis.

Coke is received as approximately −¾-in. material. Anthracite fines usually consist of buckwheat No. 5 coal or half flotation anthracite and half buckwheat No. 5.

Process Development

Sintering Ore Fines: From the time the plant was started in 1951 until June 1953, −½-in. and −⅜-in. ore fines were treated exclusively. Table II shows that during the second year, operating time was increased from 84 pct to 92.5 pct, due to improved techniques. Production rate was increased from 2.2 to 2.6* tons per sq ft per day.

The ores vary markedly in sintering characteristics from one stratum to another within a mine, and even along a mining bench. Because of this, variations in sinter-plant performance on fines are common—for example, the silica content has varied between 1.5 and 16 pct during a single day.

Sintering of Taconite Concentrates: Oliver taconite concentrates were first sintered as a mixture with ore fines. Mixtures containing less than 50 pct taconite produced fair quality sinter at the rate of 1200 tpd, considerably less than the 1500 ton average obtained when treating ore fines alone.

First trials with 100 pct taconite concentrates were unsuccessful. Plugging of chutes, pugmills, and the swing spout feeder, along with poor bed permeability, were serious problems. Widening and steepening chutes and eliminating baffles assured a fairly steady feed to the machine, but production in excess of 700 to 800 tpd was unlikely with the existing facilities, which provided limited quantities of −⅝-in. hot returns and −¼-in. cold returns. Unburned material was discharged, hot returns were of poor quality, and there was severe dusting at the discharge end. Overall operation was extremely erratic.

Experiments with Cold Returns: Following reports by Elliot[2] at Appleby Frodingham, it was thought that having a large supply of sized cold returns, which could be used to supplement the hot returns, would provide a means of improving bed permeability and increasing production. An additional plow was installed in the cooler, and other changes were made that allowed −2-in. sinter to be returned to the plant and crushed to the desired size.

By using a total of 48 pct returns (24 pct hot and 24 pct −¾-in. cold) and a 6-in. bed depth, a rate of 1000 tpd of good-quality sinter could be sustained; however, because of the high circulating load of returns, fuel consumption rose from 2.0 to 2.7 million Btu per ton of product.

Limestone and Slaked Lime Additions: Hessle[3] and Stanley and Mead[4] reported that additions of lime improved the sintering rates of fine materials, and it was decided to try this as an addition to the sinter feed. The limestone had a striking effect on the permeability of the sinter bed. As the limestone additive reached the sinter machine, windbox drafts dropped from 24 in. to 21 in. w.g. As sintering continued, drafts dropped to 17½ in. w.g., and the temperature in No. 13 windbox rose from 500° to 1000°F.

Good-quality sinter was produced at the rate of 1200 tpd using 5 pct limestone, 33 pct total returns, and a bed depth of 7½ in. to 8 in. Total fuel consumption was 2.0 million Btu per ton of product. Improvement in fuel consumption is attributed largely to the 48 pct to 33 pct decrease in the returns required. The hot-returns bin was altered and a table feeder was installed, which provided accurate control over the feed rate of returns.

Following the excellent improvement obtained with the limestone additions, it was decided to test the effect of slaked lime. Sintering trials were made adding slaked lime to the machine feed in the amount of 0.5 to 1.0 pct by weight of the taconite. The results obtained duplicated those attained with limestone additive, with the additional advantages of slightly lower fuel consumption and greater sinter strength.

* Throughout this paper, tons refers to long tons.

R. L. BENNETT, R. E. HAGEN, and M. V. MIELKE are with the Oliver Mining Div., U. S. Steel Corp., Duluth. Paper presented at the Blast Furnace, Coke Oven, and Raw Materials Conference, April 1956, Cincinnati.

Table I. Properties of Raw Materials

	Ore Fines	Taconite Conc.[a]	Coke as Received	Ground Coke	Anthracite Fines
Chemical Analyses, Pct by Wt, Dry Basis					
Fe	60.50	63.23			
SiO_2	5.75	8.12			
CaO[b]		0.90			71.00
O and V	5.89				
P	0.070	0.014			
Mn	0.38	0.24			
H_2O	13.51	10.08		10.94	11.51
Total C				77.52	78.30
Ash				13.97	14.41
Btu per lb				12,840	13,318
Sieve Analyses, Cumulative Pct Retained					
0.742 in			3.1		
0.525 in			15.0		
0.371 in	5.9		31.8		
3 mesh	12.3		43.3	0.2	0.2
8 mesh	27.8		65.3	9.1	4.4
20 mesh	38.6		79.2	36.2	26.9
65 mesh	64.6		91.8	70.9	91.9
100 mesh	70.7			78.9	95.4

[a] Taconite concentrates treated in the sinter plant are approximately 80 pct −200 mesh.
[b] Hydrated lime is essentially all −200 mesh and contains 71 pct CaO.

Hot Returns versus Cold Returns: Additional trials were made to improve the sintering of taconite concentrates using hydrated lime. Previous tests on ore fines, in which all returns were hot in one case and cold in the other, had shown that an increase in capacity was obtained with the hot returns. These results prompted tests in which only hot returns were used. Openings in the hot-returns grizzly were increased from $\frac{5}{16}$-in. to $\frac{7}{16}$-in. This made hot returns available in quantities up to 30 pct of the machine feed, improved the size consist, increased the overall temperature, and provided a better hearth layer, since the larger return particles tend to roll to the bottom of the bed at the feed point.

Table II. Process Data

Season	Long Wet Tons Ore	Long Wet Tons Sinter	Pct of Recovery	Operating Hours	Pct of Operating Time	Av Tons per Day	Av Tons per Sq Ft per Day
1951	267,709	226,447	84.0	4332	84.0	1260	2.18
1952	472,331	405,407	86.0	6564	92.5	1490	2.58

Summary: Production rates between 1250 and 1550 tpd at a total fuel consumption of 1.8 million Btu per ton are now readily sustained when treating 100 pct taconite concentrate. The following are considered to be the major factors that have contributed to doubling the production rate of taconite sinter:

1. mechanical alterations that allow steady, controlled feeding of raw materials and return fines to the machine;
2. a greater supply of hot returns of improved quality, i.e., larger size and higher temperature;
3. addition of hydrated lime to the machine feed in the amount of 0.5 to 1 pct of the taconite concentrate.

Advantages gained from mechanical alterations

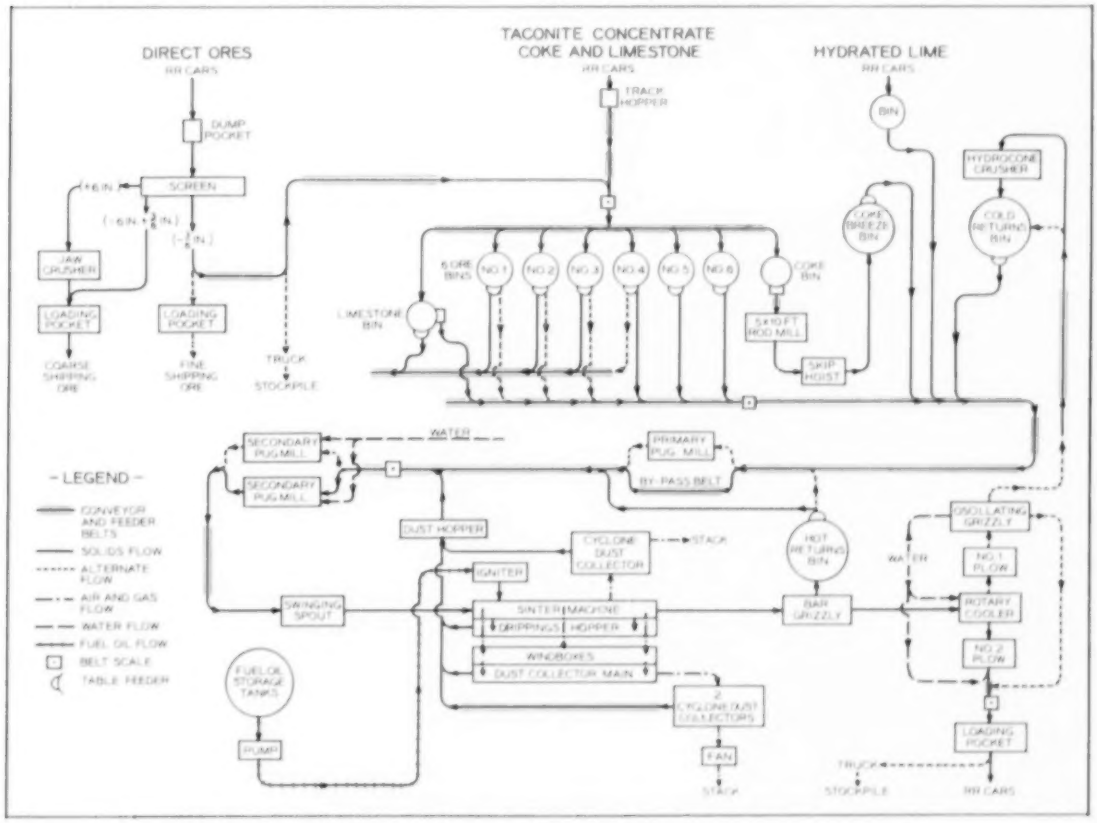


Fig. 1—Flowsheet of the Extaca plant for sintering iron ore fines and taconite concentrates.

lie in feed uniformity and bed permeability. In providing a greater supply of hot returns, the improved hearth layer reduced the circulating dust load from 1 to 0.75 tons per hr, and reduced the dust load passing through the fan, thereby increasing fan life. In addition, the larger size of the hot returns contributed to improved bed permeability. The higher temperature of the hot returns (incandescent for the most part) is due partly to the larger grizzly openings, which admit particles of good sinter from the lower and hot portion of the bed, and partly to the improved bed permeability attained as a result of all three of the factors listed. Complete burning through of the bed results, thus eliminating fine, cold, unburned material from the hot-returns. The net effect was to raise the bed temperature under the swinging spout from 100°F to 150°F.

Improved permeability and burning is explained as follows: Water is added to a sinter mix causing fine particles to form aggregates, perhaps 1/16-in. to ⅛-in. in size. Added water tends to condense in the lower, cooler part of the bed during the early stages of sintering and decrease bed permeability. Optimum water content is therefore a compromise which will give the greatest aggregation without destroying permeability through condensation.

The mechanism by which permeability increases when lime additions are used is not thoroughly understood. The size of the limestone is about 50 pct +28-mesh—coarse enough to give some increase in permeability. In addition, CaO forms low-melting calcium silicates and ferrites during the burning operation. The greater fluidity of these compounds may result in less resistance to air flow, and it is possible that bed permeability may be increased by the release of CO_2 as the burning zone advances.

Improved burning conditions obtained with the use of slaked lime are believed to be partly due to the fluxing effect of the CaO, but primarily to increased aggregation of magnetite particles. This is apparent from visual inspection of the feed and from bulk-density measurements. Bulk densities from two determinations made on the wet feed when slaked lime was used were 116.3 and 115.2 lb per cu ft, as compared with 120.7 and 121.0 lb per cu ft when limestone was used. Also, moisture content of the feed should be less when slaked lime is used.

Sinter Quality: Some of the properties of sinters made at Extaca are shown in Table III. The strongest sinter, made from taconite concentrates with no additives, was a rather solidly fused mass, whereas the lime sinter has a more scoriated, macroporous structure. Even so, the former has slightly more microporosity than the latter; it appears that the fluxing effect of the lime reduces the microporosity of the sinter.

Taconite sinter made with limestone has a strength index[c] of about 1.3, and with slaked lime it is about 2.0. The poorer strength is attributed, in part, to particles of calcined but unslagged limestone.

The size of taconite sinter is about 60 pct +½-in. and over 90 pct +10-mesh as produced (samples taken from cooler discharge conveyor).

Special Tests: From measurements of the CO_2 content in the exhaust gases and from air-flow estimates made by anemometer over the bed, the amount of air leakage into the exhaust system was calculated. It was estimated that leakage through the pallet seals plus side-wall leakage, and leakage through flapper valves on the dust legs amounted to 25.7 pct of the gas flow in the fan evase.

In exploring possible alternative fuels, tests were made in which some 500 tons of petroleum coke were consumed. It was concluded that petroleum coke is a satisfactory alternate fuel, although anthracite fines or coke breeze are preferred.

Table III. Physical and Chemical Quality of Sinter

Sinter Feed	Strength Index[a]	Micro-porosity[b]
Ore fines	0.9-1.7 Av. 1.4	13.5-17.1 Av. 15.0
Taconite concentrate, no additives	2.2-4.6 Av. 2.9	12.5-18.0 Av. 15.2
Taconite concentrate plus 5 pct limestone	0.8-1.5 Av. 1.3	10.4-15.0 Av. 13.0
Taconite concentrate plus 0.7 pct slaked lime	1.4-2.2 Av. 2.0	Av. 14.3

Constituent	Chemical Analyses, Pct by Wt, Dry Basis: Group 3 Fines Sinter	Chemical Analyses, Pct by Wt, Dry Basis: Taconite Conc. Sinter	Typical Size Consist of Extaca Sinter: Mesh	Taconite Conc. (Slaked Lime Additive), Cum. Pct Ret.	Group 3 Ore Fines, Cum. Pct Ret.
Fe	64.70	63.18	2 in.	10.0	18.7
SiO_2	6.34	8.10	1 in.	33.0	30.1
CaO	0.15	1.50	½ in.	61.5	54.0
P	0.060	0.014	3 mesh	75.9	74.9
Mn	0.53	0.23	10 mesh	91.6	93.1
H_2O	1.58	1.57	65 mesh	98.4	98.9
Total C	0.091	0.153	100 mesh	98.7	99.2
Al_2O_3	1.67	0.86			
S	0.005	0.005			

[a] Pct of original particle volume after tumbling 2500 grams for 200 revolutions in a standard coke tumbling drum.[5]
[b] Pct −14-micron pores.

Operation

A flowsheet of the Extaca sintering plant is shown in Fig. 1. The plant is operated by four men and a plant foreman on each shift. A burnerman and his assistant are responsible for the sinter machine and control of the burning operation. A table feederman keeps the ore, fuel, and lime feeding from the bins properly, and a sinter-return man is responsible for proper functioning of windleg flapper valves, dust-bin feeder, hot-returns feeder, sinter cooler, and product conveyors.

The sintering machine is operated to give maximum production of good-quality sinter, that is, the cake is discharged as soon as the bed is burned through. The operation is controlled from the discharge end. By observing the sinter cake the burnerman decides whether to make changes in fuel and feed rates. Windbox drafts and temperatures are helpful in controlling the operation, but the character of the bed at discharge is the determining factor.

Moisture is changed by a manually operated gate valve that controls water flow to the secondary pugmills. Most operators judge moisture content of the feed by grabbing a handful and observing the amount of aggregation of the fine particles. Again, the windbox draft indicators are helpful in achieving optimum water addition.

The quantity of hot returns available depends on the grizzly opening at the discharge and the quality of the sinter. The 50-ton refractory-lined hot-returns bin allows use of increased amounts of hot returns for short periods to recover from upsets.

Extreme care is exercised to prevent the discharge of unburned material, which would find its way into the returns bin and cause deterioration in the quality of the hot returns.

Proportioning of ore and fuel is accomplished by adjustment of gates on the table feeders. Once the desired proportion of ore and fuel is attained, the feed may be adjusted from the burner floor by means of a rheostat control, which speeds up or slows down all feeders simultaneously while maintaining the set proportion of ore and fuel.

There has been very little difficulty in maintaining controlled rates of feed from the table feeders, and troublesome stoppages at bins have become

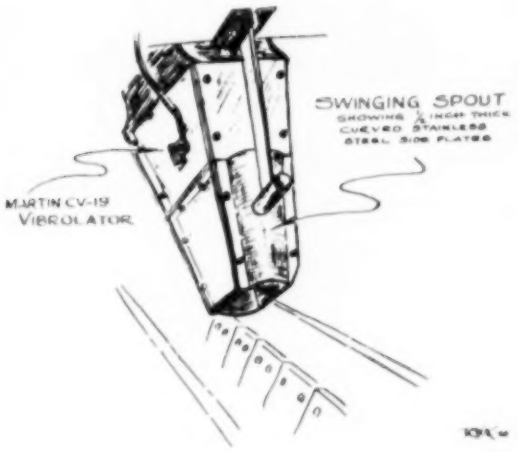


Fig. 2—The swinging spout, with modifications for easy flow.

infrequent since installation of pneumatic vibrators. Despite the fact that frost chunks up to 12 in. can be loaded into the bins, they are in a heated section of the building; so, there is no tendency to build up large frost pillars.

Coke Rod Mill

The coke rod mill is equipped with Lorain liners. A normal load is 21,000 lb of 3½-in. diam high carbon steel grinding rods, equivalent to 45 pct rod load by volume.

The high moisture content of the coke introduced several problems. Plugging of the feed spout was overcome by using a screw feeder. Condensation at the peripheral discharge ports caused plugging, which was largely eliminated by blowing air in with the feed and removing the housing over the discharge ports.

After making these changes, it was possible to grind up to 7 tons per hr of ¾-in. coke to −10-mesh. Steel consumption, including scrap loss, is now 0.919 lb rods and 0.182 lb liners per wet ton of coke. Average energy consumption is 5.3 kw-hr per ton.

Pugmills

The original flame-hardened cast-steel pugmill paddles, and manganese-steel paddles had to be discarded after about 175 hr. Cast Ni-Hard was tested but the thicker section required for strength caused plugging.

Hard-surfaced welding has helped markedly in the maintenance of pugmills. Although any surface above 500 Brinell is satisfactory, practice at Extaca is to use Stoody Borod on the paddle tips and Self-Hardening on the sides. Ease and method of applying hard surfacing is an important factor in choosing a type of rod.

Paddle life averages 1000 to 2000 hr by resurfacing in place after each 200 to 300 hr of service.

Tungsten carbide inserts show no improvement over hard surfacing and have the disadvantage of possible loss and damage from tramp iron.

Longer paddle life and lower power consumption were obtained by mounting paddles at 90° to the shaft instead of 60° (small angle toward feed end). No deleterious effect on mixing resulted from this change.

Swinging Spout

Plugging of the swinging spout, Fig. 2, by sticky, fine ore, particularly wet taconite concentrate, was a serious obstacle to steady feeding. Plugging is now minimized by using outwardly curved sides (16-in. radius) of stainless steel. Sticky material does not adhere as readily to the polished surfaces or to the rounded corners. Swinging hammers rap the spout continuously on each side as it swings. The sides of the spout can be heated by oil burners which helps flow. The original rubber lining on the sides was removed because of excessive maintenance.

Fluffer and Hearth Layer: To reduce the amount of material drawn through the grates, a hearth layer is put down. This is accomplished by replacing the fluffer grid with a sloping shaking bar grizzly, which is arranged so that coarser portions of the feed are screened out and thrown down first on the grate bars.

Sinter Machine

Ignition Furnace: The ignition furnace was originally equipped with six fuel-oil burners. To reduce radiant heat loss, improve ignition and minimize local glazing, the furnace was lowered as much as possible at the time the pallet sides were cut from 14 in. to 10 in. Another burner was added at the back wall, and all burners were directed downward at a shallow angle instead of impinging directly upon a small spot of the bed. These changes reduced ignition fuel consumption about 20 pct.

Grate Bars: Grate-bar replacements average 12.1 per day, equivalent to 0.08 lb per ton of sinter shipped. Grate bars now used are a compromise between gray and white cast iron and contain about 0.5 pct Ni and 1.5 to 2.0 pct Cr. Bars used previously included 1½-in. Mallix bars and various gray and white iron alloys, but all these warped excessively. Bars of white cast iron were too brittle and excessive breakage occurred.

Sinter Machine Pallets: On most pallets, at least two of the four longitudinal members cracked during the first six months of operation. The cracks occurred through the heaviest sections. Failure was attributed to shrinkage cracks, too wide an opening between pallets at the machine discharge, or grate bars placed too tightly on the pallets. Redesigning the pallets to have lighter cross members and substituting ductile iron for cast iron has increased service life by 4 or 5 years. Pallet side plates were cut from 14 in. to 10 in. to accommodate the shallow beds of taconite concentrate.

Sinter Machine Pallet Wheels: Average life of pallet wheel bearings is only 8 months, and steps to improve it have not been particularly successful. Grease seals of various designs; materials such as neoprene, felt, leather, silicone, rubber; and a wide variety of high-temperature lubricants, have made no substantial improvement. Present practice is to

grease every other day in an attempt to force out dust-contaminated grease. The source of trouble is believed to be the *breathing* of dust, which takes place with each cycle of heating and cooling.

Cooler

The cooler, Fig. 3, is the Fontana type, consisting of a rotating bin of annular shape. Hot sinter is loaded at the top and plowed out at the bottom. It was intended that cooling should take place entirely by radiation and natural draft. But when cooling by these means, sinter discharges at about 400°F, with individual pieces still red hot. To cool this material further, water is sprayed over the discharge from the cooler and on the cooler itself just before the point where hot sinter is added. Water consumption is 35 to 50 gal per ton of sinter.

Sinter Return System

The structural members that support the hot-fines grizzly had to be replaced by air-cooled box girders to overcome deformation resulting from heat and the impact of sinter slabs dropping off the end of the machine. Now, before sliding over the grizzly, the sinter falls on an impact plate, which is divided into two 6-ft sections. Grizzly bars are made of Ni-Hard, which resists abrasion well but tends to warp. The upper grizzly section is subjected to some impact from discharged sinter and has a service life of about one year, whereas the lower grizzly, which is subjected to abrasion only, has a life of about two years.

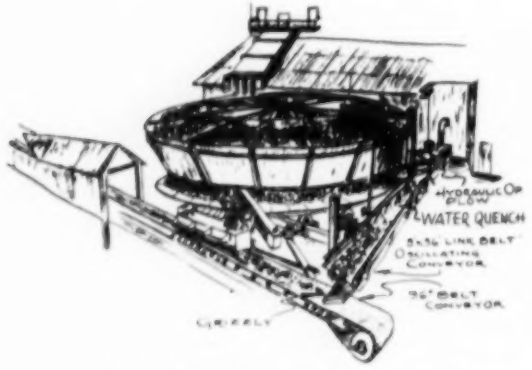


Fig. 3—Fontana-type rotary sinter cooler.

Mechanical difficulties in feeding hot returns by the original vibrating feeder have been largely overcome by replacing it with a table feeder of 6-ft 6-in. diam. This feeder was first equipped with a deck of Ni-Hard wear plates, which lasted about three months. The present 3-in. deck of silicon carbide brick has been in service 21 months without appreciable wear. The hot-returns bin above the table feeder was rebuilt and lined with first-quality firebrick which is replaced annually. Testing of silicon carbide brick lining is underway.

A vibrating feeder with grizzly section was originally used to convey sinter from the cooler to No. 9 belt, the grizzly undersize providing a source of cold returns. As this installation was mechanically unsatisfactory, it was replaced by an oscillating trough conveyor with a 3-ft long grizzly section. At the present time, cold returns are seldom used, and the entire sinter product is plowed directly upon No. 9 belt from the cooler.

Product Handling

The product conveyor belt is subject to extreme abuse from coarse abrasive material and high temperatures. Original belt life was only 8 to 10 months. After two belt changes conditions were improved somewhat by loading over a grizzly, bedding the large hot chunks with cooler fines. Since a belt tends to wear out each year, a relatively low-cost 36-in. belt was installed.

Sinter Fan

Changes were made to the sinter fan to increase air flow and to improve wear. The original 109-in. rotor was replaced by one of 116-in. diam. The fan will move 134,000 cfm at a fan static pressure of 25.5 in. w.g. and a temperature of 300°F. The original fan rotor with mild steel wear pads lasted for one year while sintering ore fines. A rotor with $\frac{3}{8}$-in. thick high-carbon wear pads also lasted one year, but during 20 pct of this time, the plant treated taconite concentrates, which cause greater dusting and abrasion. A redesigned rotor in which the entire blade surface is replaceable has been in service since April 1955. Diamond floor-plate wear pads, $\frac{3}{8}$-in. thick, are used on this rotor, and a service life in excess of one year is expected.

This fan was designed with antifriction bearings, which tend to overheat in the summer. A water-cooled babbitt bearing would be preferable and could be cooled by circulating transformer oil in the winter.

Cyclone Dust Collectors

The lower part of the cylindrical section is subjected to excessive abrasion; so, it was lined with silicon carbide bricks. A 5-ft 9-in. diam central discharge pipe originally extended to within one foot of the bottom of the cylindrical section. Shortening this pipe by 12 in. decreased the pressure drop through the cyclone by $\frac{3}{4}$ in. w. g. and it is believed that collection efficiency has been improved. As a further step to improve efficiency, the conventional flapper valves on the dust legs have been replaced by Buell positive seal mechanical dust valves.

Instrumentation and Control

The entire sinter plant is operated from a central control room located near the feed end of the sinter machine. The control panel and instruments are housed in a pressurized room, with operating controls on the outside wall of the room in front of the panel. The following controls are provided: 1) Fan damper and position indicator; 2) Sinter machine speed control with recorder; 3) Cooler speed indicator and recorder; 4) Windbox temperature and draft indicator and recorder; 5) Pugmill water indicator; 6) Equipment start-stop buttons and running lights; 7) Fan-power meter and pugmill ammeters; and 8) Flow-rate recorder (tons per hr) of sinter machine feed before and after hot returns are added.

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The Krupp-Renn Process in Czechoslovakia

Based on an article

submitted by the Czechoslovakian Delegation to the Steel Committee of the Economic Commission for Europe, and

Published in

Advances in Steel Technology in 1955 by the United Nations in Geneva, Switzerland

IN Czechoslovakia there are deposits of highly acid iron ores, averaging some 28 pct Fe and 25 pct SiO_2. The alumina content is some 8 pct, while basic components, such as CaO and MgO, are found only in negligible quantities. Because of the excessive quantities of coke and limestone required, such ores cannot be economically treated by traditional blast furnace technique.

Numerous beneficiation experiments have been conducted on these ores, but due to their strongly striated structure, with the iron and silica chemically combined, the results have been poor. To process these ores a Krupp-Renn installation was, thus, laid down in Czechoslovakia. It does not differ greatly from plants previously constructed in Korea, Japan, and at Watenstedt in Western Germany.

A brief outline of the process is as follows: Ore, reduced to 25 mm, is blended to 27 pct of its weight with powdered coke of 5 mm or less. These are mixed into a homogeneous mass and charged into the Krupp-Renn furnace, a rotary kiln, 3-m interior diam and 60-m long. It revolves once every 52 seconds. The furnace is fired with pulverized coal with a heating value of 6000 Kcal. Fuel requirements amount to some 7 pct of the weight of the charge. The burners are located at the exit end of the furnace where the temperature is maintained between 1300° and 1400°C. The combustion gases traverse the furnace and leave at the charging end to enter a dust collector at about 450°C.

The charge moves through the furnace by means of the rotation and the 2 pct inclination. It is dried and preheated to 500°C by the time that it has traversed one third of the furnace. At this point reduction commences and continues until the ore is discharged; however, the greatest reduction takes place in the central portion of the furnace where the temperature is between 600° and 1000°C. With mounting temperature minute particles of iron form in the charge, and the mass is transformed into a pasty substance. The particles of deoxidized iron coagulate under the rotating action and appear in the form of grains of variable dimensions, but generally not exceeding 10 mm. This slaggy mixture passes out the furnace and into a pan conveyor which carries it to the storage area. The mixture of Krupp-Renn iron grains and slag cools for about two days and is then transferred by clam shell bucket to the grinding department.

The cooled slaggy mass is ground in a ball mill to liberate the grains of iron from the slag, and this is followed by screening and separation by electromagnetic cylinders. Iron grains larger than 0.8 mm are for blast furnace charging, while the finer material, which contains almost 60 pct Fe, is returned to the Krupp-Renn furnace.

The present Krupp-Renn process has, in addition to its many advantages, a certain number of serious defects, which explain why the process is not more widely used at present. The following are the advantages:

1) It does not require metallurgical coke, but instead makes use of fuel and reducing medium of inferior quality, with the exception of pulverized coal for the furnace fuel. The latter is used only in small quantity and may be replaced by other combustibles.

2) The iron-slaggy material is readily separated into iron grains containing between 88 and 90 pct Fe and less than 5 pct SiO_2.

3) The remelting and desulfurization of the grains in a blast furnace requires only 300 kg of coke per metric ton charged.

4) For acid ores the over-all cost of production of pig iron obtained by the Krupp-Renn process is well below that obtained by direct reduction in a blast furnace.

5) The Krupp-Renn process permits the utilization of ore fines without recourse to sintering.

The principal disadvantages of the process are: 1) high capital expenditure; 2) bad working conditions caused by large quantities of dust released in processing the Krupp-Renn iron grains; and 3) high production costs in spite of point 4, above.

Metallurgists are cognizant of these facts, and with the construction and operation of some new plants, they hope to gain further experience in this process. The Department of Research has already embarked on a research program designed to simplify and improve the over-all Krupp-Renn process, reduce capital expenditures, and accelerate the operation. Semi-commercial installations are being constructed, and two modified techniques have emerged from this project. The first consists of reducing the hot mass leaving the furnace to grain size by means of a current of water on the cooling belt and following this by electromagnetic separation. The second method consists of utilizing a low shaft furnace to treat the hot product of the Krupp-Renn kiln, raising the temperature of the mass until the slag and metal separate by difference in density.

In addition, studies will be undertaken regarding the best means of utilizing the gases forming in the interior of the furnace to preheat the combustion air for the burners. Finally, trials will be made on the utilization of fuels for which there is no other use.

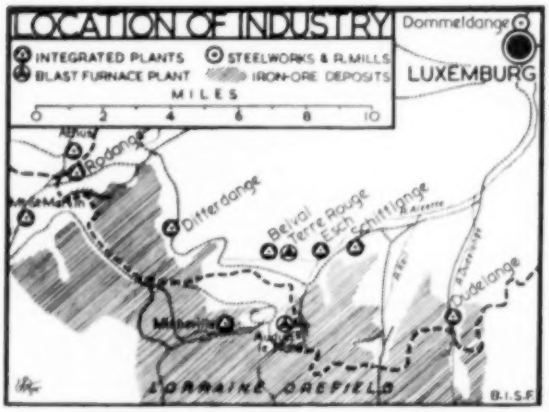
Positive results from these experiments should permit the realization of substantial progress on the road to the more economic exploitation of the Krupp-Renn process.

Agglomeration of Luxembourg and Lorraine Iron Ore Fines

Based on an article by

J. Paquet

in Advances in Steel Technology in 1955, published by the United Nations, Geneva, Switzerland, 1956



Courtesy British Iron & Steel Federation

THE steel industry of Luxembourg is based on deposits of *minette*, an oölitic limonite, averaging less than 30 pct Fe. The country possesses no supply of coking coal; hence, there is need for operating blast furnaces as efficiently as possible with the lowest coke rates.

The ARBED firm, Luxembourg's largest steel producer, operates three integrated plants in that country and one in the Saar. The latter plant, at Burbach, is adjacent to coal mines and operates its own coke ovens. There, Dwight-Lloyd machines, requiring coke fines, have been installed. Nevertheless, with coke fines becoming somewhat scarce as the sintering technique is more widely used, the Burbach plant has installed, in cooperation with the Lurgi firm, a system of mixed heating whereby almost one half of the solid combustibles are replaced by gas.

Rotating furnaces, operating on blast furnace gas, have been used since 1931 at the Beval and Dudelange plants in Luxembourg. Initially they served only for treating blast furnace flue dust, but since crushing minette ores produces from 15 to 25 pct fines of −10 mm, there has been a definite need to agglomerate fines.

After World War II, crushing, screening, and nodulation were introduced in all of the Luxembourg plants of ARBED. This firm now operates ten rotating furnaces, built by the Smidth firm of Copenhagen. There are adequate supplies of blast furnace gas for the operation of these rotating furnaces, except in winter, when there is a scarcity of fuel in the area, and liquid or solid fuels must be substituted at some power installations normally dependent upon blast furnace gases.

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Nodulation Installation at the Terres Rouges Plant in Luxembourg

This plant has four rotating furnaces, 92 m long, 3.6 m in diam at the preheating zone, and 5 m in diam at the reduction zone.

Stocks of ore fines, which may amount to as much as 20,000 metric tons, are moved by a grab chain and deposited into one of two hoppers feeding a conveying belt. This carries the fines to the feeding tower situated above the furnace. Scraper chains distribute the mineral in conical hoppers, from where it is drawn off by apron feeders and charged in the furnace by a chute.

Each furnace is driven by a 150 kw electric motor at a variable speed so that the furnace may operate between 0.4 and 1.2 rpm. The furnace is constructed of refractory lined welded steel plates and is heated by blast furnace gas. The air for combustion is pre-

heated to 500°C in reheating furnaces which are also fired by blast furnace gas.

The agglomerate, leaving the furnace, is conveyed by a metallic conveyor through a 50 m long tunnel where it is submitted to the first cooling by a counter current of air. This air, heated to about 300°C in passing through the tunnel, is used after a second heating as the combustion air in the rotating furnaces.

Two bucket transport systems receive the precooled agglomerate and transport it to two coolers of the Lurgi type. These consist of inclined metal transporters, the bottom of which is composed of louvers. These allow cooling air to be sucked across the agglomerate by three ventilators. The agglomerate is cooled to at least 120°C and then discharged into boxes in which it is taken by auto discharge wagons to the blast furnace plant.

The fumes leave the rotating furnace at about 300°C. Dust is precipitated by successive treatment in cyclones and electrostatic separators, both of Lurgi design. In leaving the smoke stacks, the fumes are completely colorless and contain less than 50 mg of dust per cu m of fume. The dust collected, amounting to some 4 tons per hr per furnace, is reintroduced into the furnace by means of compressed air.

Blast furnace flue dust is received by air conveyance in two hoppers and carried by compressed air to the feeding tower of the agglomeration unit. Introduction to the rotating furnace is made, through the intermediary of four variable speed transport screws, by the same chute utilized for the fines.

Dwight-Lloyd and Rotating Furnace Processes

Chemical Properties: Table I shows typical chemical analyses of the two types of agglomerate. Their chemical properties differ from each other with respect to the degree of iron reduction. In the case of the agglomerates produced with the addition of blast furnace flue dust, the carbon contained in the dust remains in effective contact with the charge of the rotating furnace from 3 to 5 hr, while the Dwight-Lloyd reaction time is no longer than a half hour. Actually, blast furnace practice has shown that there is no apparent difference in coke consumption when using one agglomeration process or the other.

Physical Properties: Table II shows the general density and porosity tests on the two agglomerates.

The nodules from the rotating furnace have a greater mechanical strength than the Dwight-Lloyd agglomerate. The latter is partially broken up by manipulations and successive handling before charging in the blast furnace, while the rotating furnace nodules are charged without great damage.

Production Requirements: Table III shows the production requirements for the two agglomeration techniques.

General Considerations and Conclusions

In examining the two methods of agglomeration it may be concluded that the calories necessary for the agglomeration of minette fines are practically the same in either case (about 650,000 cal for 20 pct blast furnace flue dust and 80 pct ore fines). In the Dwight-Lloyd process this heat is normally supplied by coke fines and in the rotating furnace by blast furnace gas. In the production costs, the fuel figures out to be about two thirds of the total. Economies realized in one or the other of these processses depend, therefore, on the cost of the available fuel. The plant at Burbach, Saar was constructed in proximity to large coal mines and possesses coke ovens. The ARBED plants in Luxembourg smelt low grade minettes and have no available coke fines, but by contrast, they do have a large supply of blast furnace gas, permitting the economical operation of the rotating furnaces.

Summarizing, it may be said that the two agglomeration processes differ with regard to the quality of the product. The Dwight-Lloyd sinter is more porous and more reducible than the nodules from the rotating furnace, while the latter is more resistant and contains more ferrous oxide by reason of the relatively long association between iron and carbon in the rotating furnace.

In the case of either process, the agglomerate should be distributed proportionally in each blast furnace, and its percentage should not exceed 25 to 30 pct of the charge. This conforms to the theory that agglomerates are always less reducible than the minettes from which they are made. The difference in reducibility is counter-balanced by the greater permeability of the agglomerate when the percentage of agglomerate does not exceed 30 pct of the burden.

Results have thus shown that the choice of either of these two agglomeration processes depends largely on the cost of the available fuel.

Table I. Chemical Analysis of Agglomerates

		Rotating Furnaces	
	Dwight-Lloyd, at Burbach, Saar, Using 20 Pct Dust	at Reval, Using 20 Pct Dust, Pct	at Terres Rouges, Using No Dust, Pct
Fe total	40.68	41.82	40.33
SiO_2	19.18	16.86	17.38
CaO	11.81	14.89	15.20
MgO	2.17	1.60	1.81
P_2O_5	1.72	1.76	1.79
MnO	0.77	0.46	0.30
Al_2O_3	6.64	6.67	6.38
CO_2	0.17	0.21	0.00
C	0.68	0.20	0.09
S	0.10	0.06	0.00
Distribution of iron			
Fe_2O_3	37.92	24.71	51.19
FeO	18.70	31.08	5.24
Fe in Fe_2O_3	25.89	17.28	35.82
Fe in FeO	14.52	24.16	4.08
Fe, metallic	0.27	0.38	0.43
Total iron	40.68	41.82	40.33

Table II. Porosity and Density Data on Dwight-Lloyd and Rotating Furnace Agglomerates

	Porosity			Density	
	Apparent	Closed	Total	In Bulk	Absolute
Dwight-Lloyd	21.5	14.9	36.4	2.49	3.91
Rotating furnace	14.6	8.4	23.0	3.07	3.98

Table III. Production Requirements Per Ton of Agglomerate Produced by the Dwight-Lloyd and Rotating Furnace Techniques

	Rotating Furnace	Dwight-Lloyd
Coke fines	—	67 kg
Blast furnace gas	650 cu m	205 cu m
Steam	—	4.70 kg
Energy	12 kw	25 kw
Water	5 kg	3.15 kg
Labor (incl. unloading)	0.35 hr	0.35 hr
Compressed air	13 cu m	10 cu m
Maintenance	12 pct of total costs	

Self-Fluxing Agglomerates in the USSR

Based on an article

submitted by the Soviet Delegation to the Steel Committee of the United Nations' Economic Commission for Europe, appearing in Advances in Steel Technology in 1955, published by the United Nations, Geneva, Switzerland, 1956.

IN Russia, two steel plants are operating blast furnaces on burdens containing from 75 to 90 pct agglomerates. In order to improve the operation it was decided to add flux, in the form of finely ground limestone, to the agglomerate. In this manner, the decomposition of the limestone does not take place in the blast furnace but rather on the agglomeration strand, thereby substantially reducing the blast furnace coke rate and increasing output.

At plant A, the limestone is received in a size range of 10 mm to 100 mm, ground in a hammer mill, and screened in a closed circuit to −4 mm. At plant B, a ball mill is employed for dry grinding the limestone to between 85 and 90 pct −3 mm.

At plant A, between 16 and 18 pct limestone is added to the agglomeration mixture to produce an agglomerate with average basicity ($CaO:SiO_2$) of 1.3. During agglomeration, intimate contact is established between the limestone and other constituents, resulting in an agglomerate of satisfactory mechanical strength. The elimination of sulfur is not decreased by limestone additions. Moreover, the introduction of limestone renders the agglomerate more reducible in the blast furnace, for such lime agglomerate contains less iron chemically combined with silica.

Table I shows the composition of lime and nonlime agglomerates at plant A, while Table II has data on agglomeration mixes at plant B. It should be noted that increasing the quantity of limestone in plant B resulted in considerably increased production rates on the agglomeration bed.

Table III shows the blast furnace operation resulting from varying quantities of lime agglomerate in the burden at plant A. With increasing quantities the coke rate decreases and output increases constantly over that obtained with ordinary agglomerate in the charge. With basicity of 1.26 and using 86.4 pct lime agglomerate in the burden, the coke rate is 11.1 pct less and the output 12.2 pct more than when charging with ordinary agglomerate. In spite of increasing quantities of CaO instead of $CaCO_3$ in the burden, the CO_2 at the furnace throat shows a slight increase with higher lime agglomerate burdens, indicating increasing indirect reduction of ore by furnace gases.

Table IV shows the results obtained from operating with ordinary and with lime agglomerates at plant B. By using some 63 pct lime agglomerate of 0.6 basicity, coke rates were reduced by about 6 pct as compared with ordinary agglomerate burden operation, and at the same time output was increased by as much as 5.3 pct.

The experimental burdening of blast furnaces at these two plants shows that the use of lime agglomerate results in a reduction of coke rates and an increase in the rate of blast furnace output.

Table I. Composition of Lime and Nonlime Agglomerates at Plant A

Agglomerate Type	Composition, Pct							
	Fe	FeO	SiO_2*	CaO	MgO	Al_2O_3	S	SiO_2*
High S magnetite concentrate, limestone added	55.2	16.4	7.81	10.33	1.86	2.52	0.040	1.32
High S magnetite concentrate, no limestone	61.2	23.6	8.45	2.10	0.65	2.55	0.036	0.25
High S oxide fines, concentrate, limestone added	52.2	17.0	9.34	12.55	1.24	3.60	0.043	1.34
High S oxide fines, concentrate, no limestone	58.3	23.2	10.32	2.96	0.6	4.49	0.027	0.29

* It is assumed that the silica in the column at left is chemically combined with iron, while the silica in the column at right is free.

Table II. Agglomeration Characteristics of Lime and Nonlime Agglomeration Mixes at Plant B

	Ordinary Agglomerate	Lime Agglomerate
Basicity ($CaO:SiO_2$)	0.12	0.6
Analysis in pct: Fe	61.0	57.5
FeO	21.2	11.6
CaO	1.4	6.5
Size analysis at mixers		
>25 mm	29.8	27.6
25 to 12 mm	23.7	17.7
12 to 5 mm	23.6	27.7
<5 mm	22.9	27.0
Production of agglomeration band, tons per hr	70.0	80.3

Table III. Blast Furnace Operation at Plant A with Normal and Lime Agglomerates

	Normal Agglomerate	Lime Agglomerates					Total Change, First to Last Example
Total agglomerate in burden, pct	64.5	71.7	71.5	78.8	94.4	86.4	
Lime agglomerate in burden, pct	—	36.6	36.8	78.8	94.4	86.4	
Basicity of agglomerate	—	0.75	0.80	0.78	0.87	1.26	
Coke rate decrease, pct	—	3.0	—	3.4	—	4.7	11.1
CO_2 at throat of blast furnace	11.0	10.9	11.7	12.1	12.8	12.9	
Increase in output, pct	—	5.5	—	3.0	—	3.7	12.2

Table IV. Operating Results of Blast Furnaces One and Two, Plant B, with Normal and Lime Agglomerates

	Blast Furnace One		Blast Furnace Two	
Total agglomerate in burden, pct	75.7	74.1	73.6	72.0
Lime agglomerate in burden, pct	—	64.4	—	62.9
Fe in burden, pct	44.2	44.5	44.1	43.2
Coke rate, kg per ton iron	867	815	864	814
Dust at furnace throat, kg per ton iron	211	152	129	81

Journal of Metals Award Paper - - 1956

Reaction Zones in the Iron Ore Sintering Process

by R. D. Burlingame, Gust Bitsianes, and T. L. Joseph

DESPITE almost fifty years of commercial practice, the sintering of iron ore has received little fundamental study. Much of the theoretical work[1-4] has dealt with the constitution of sinter produced under widely varying conditions. While these studies have broadened our knowledge of the changes that occur in the sintering zone and in the freshly formed sinter during the early stages of cooling, they provide little insight into the changes that precede the formation of sinter. These preliminary changes merit study as a part of the overall process.

Hessle,[5] working with beds of Swedish magnetite concentrates, was one of the first investigators to study the sintering process in its entirety. On the basis of temperatures observed at various levels of the bed during sintering, he postulated a number of distinct reaction zones to account for the chemical changes leading to the formation of sinter.

A more direct method of attack is that of arresting the sintering zone after it has progressed part way through the bed. A study of a vertical cross section through such a *quenched* bed provides direct information on the changes taking place at various levels. This method was used by McBriar et al.[6] to show that several well-defined zones of chemical change existed within beds that were typical of British sintering practice. The same general method of attack was developed independently in the present investigation to study partially sintered beds typical of American practice.

Experimental Sintering Equipment

The sintering operation was carried out on an experimental scale with the equipment shown in Fig. 1. The refractory-walled sintering chamber *A* was 11 in. deep and averaged 9 in. in diameter. Air was introduced through a tapered flow section *B*, which contained the orifice *C* for accurate metering of the incoming air. This section was located directly above the square ignition housing *D*, which in turn rested upon the sintering chamber *A*. The bed was ignited with burner *E*.

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The required suction for the operation was furnished by a fan *F*, which had an air capacity of 500 cfm (stp). Hot exhaust gases from the sintering chamber were cleaned in the dustcatcher *G* before entering the exhaust fan.

In the study of partially sintered beds, it was essential to find some technique for removing the entire charge from the sintering pot without disarranging the unsintered bottom portion. This problem was finally solved by sintering the charge in a removable basket, which snugly fitted the sintering chamber. This basket was constructed of two thicknesses of window screen and was lined with a 3/16-in. layer of asbestos paper. The bottom of the basket consisted of two thicknesses of wire screen, which were fastened to the basket wall. For high fuel mixtures, additional insulation was provided by a somewhat thicker layer of asbestos cement.

Preparation of Partially Sintered Mixtures

The moist feed was carefully placed in the sintering basket, to prevent segregation of the particles, which varied widely in size and composition. A thermocouple was placed in the center of the basket with the hot junction halfway down, and the mixture was evenly distributed around it.

During ignition and throughout the sintering of the upper half of the bed, the hot junction temperature increased very little. When the sintering zone reached the halfway point, as indicated by the sudden increase in the hot junction temperature, the charge was quenched. During quenching the suction was turned off and the orifice was tightly stoppered to prevent further influx of air. At the same time, nitrogen was admitted to the sintering chamber through the orifice tap. As soon as the nitrogen had displaced the air and products of combustion, the charge was removed from the sintering pot for immediate dissection.

It is impossible to preserve the exact zone structure of the bed at the instant that combustion is arrested unless the downward transmission of heat is also immediately stopped. Fortunately, heat transfer is very slow in beds containing a stationary fluid, especially if the particle size is small. It follows that the minimum quantity of nitrogen should be used to displace the air and that static conditions be established as soon as possible. A very steep temperature gradient across the combustion zone for some time after the quench was evidence of in-

The Transactions pages appearing in the Journal of Metals during 1956 will subsequently appear in AIME Transactions Volume 206, and may be permanently referenced to that volume.

significant heat transfer. After completion of the dissection, the bright orange color of the hot sinter was found to grade sharply into a black zone of unfused particles of magnetite over a distance of only ½ in. The zone samples and zone dimensions, therefore, represent the conditions as they existed during sintering.

Dissection Procedures

Fig. 2 is a pictorial explanation of the two methods of dissection. One scheme proceeded according to sketches 1-2-3-4, while the other one followed the sequence 1-2*A*-3*A*-4*A*-4. The former method was the simpler, but the latter procedure also provided an opportunity to photograph the zone structure.

Sketch 1 of Fig. 2 represents the basket containing the hot, partially sintered charge that has just been lifted out of the sintering pot. The charge is inverted and the basket is then cut along the dotted line, sketch 2. Successive horizontal layers of bed are removed by alternately cutting the basket and striking off the samples or layers, sketch 3. Each horizontal section or sample is placed in a Pyrex flask, sketch 4.

In the second method of dissection, the as-quenched charge is laid on its side and the wire basket is cut along the dotted line as shown in sketch 2*A*. This exposes a 120° sector of the bed below the combustion zone. Starting at the bottom of the charge, successive slices of the unsintered bed are obtained by driving sampling blades into the material as indicated by the dotted lines of sketch 3*A*. Each sample is caught in a sampling trough, sketch 4*A*, and then transferred to a storage flask, sketch 4. The unexcavated bed clearly reveals the zone structure. To avoid oxidation, the hot particles of magnetite adjacent to the combustion zone are swept with nitrogen when removed from the bed.

Resin Impregnation and Sectioning

Samples from various levels of the unsintered portion of the charge revealed the changes occurring ahead of the combustion zone, but the conversion of the loose granular material into sinter in the very narrow combustion zone could not be studied by this technique alone. In order to prepare polished specimens from the sintering zone for microscopy, the granular bottom portion had to be bound coherently to the sinter cake. This was accomplished by impregnating the entire charge with hard-setting resins, by a specialized procedure.

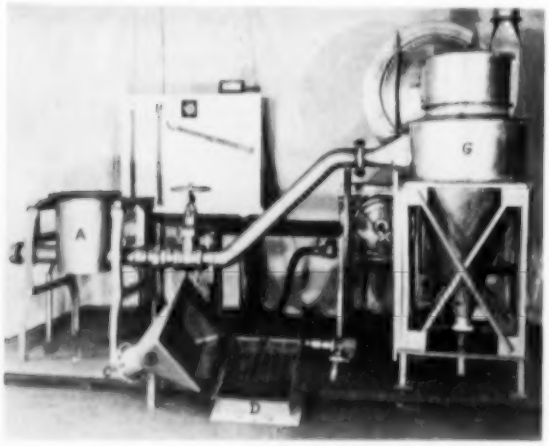


Fig. 1—Experimental sintering machine: *A*, sintering chamber; *B*, flow section; *C*, orifice for metering incoming air; *D*, ignition housing; *E*, burner; *F*, fan; and *G*, dustcatcher.

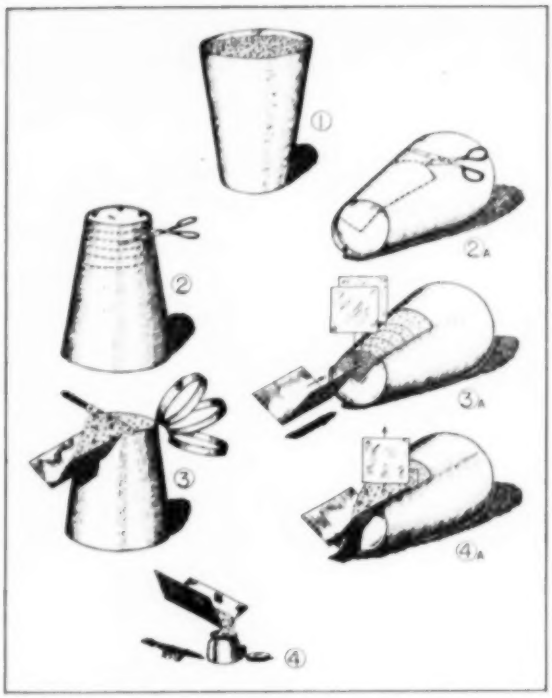


Fig. 2—Dissection procedures for partially sintered charges.

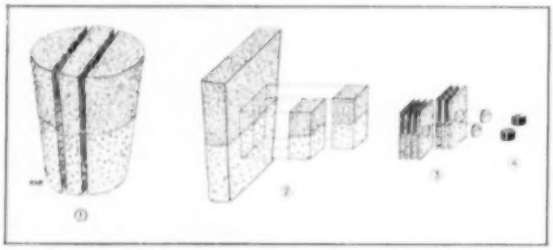


Fig. 3—Sampling procedures for resin-impregnated charges: 1) Bakelite-filled charge sawed vertically, 2) center slab of sketch 1 sawed into columns, 3) reimpregnated columns sawed into slices 1/16 in. thick, and 4) 1-in. disks for work at higher magnifications.

After a partially sintered charge had been cooled and dried in nitrogen, it was transferred to a sheet metal impregnating vessel of slightly larger dimensions. A hose connected the bottom of this vessel to a reservoir filled with a solution of 70 pct Bakelite and 30 pct acetone by weight. By slowly raising the reservoir, the liquid thoroughly filled the charge of sinter by upward percolation. After a period of soaking, the excess solution was drained off. A tough resin bond was produced by curing the charge at 150°F for at least 8 hr. However, reimpregnation was always necessary before sufficient strength was developed for good sectioning.

Sectioning was performed on a large diamond saw. Sketch 1 of Fig. 3 shows the Bakelite-filled charge after being sawed into three vertical sections. The porous sinter is on top and the impregnated granular or unsintered portion of the charge is on the bottom. The center slab was in turn sectioned into short columns, which contain the combustion zone, sketch 2, Fig. 3.

Table I. Screen Analyses of Raw Materials

Size Fraction	Wt Pct of Size Fraction					Typical Sinter Mixture,* Pct	
	Ore Fines	Flue Dust	Sinter Returns	Roll Scale	Coke Breeze	Dry State	After Wet Mixing
Inch							
−1.05 + 0.74				1.3			
+0.53			2.5	2.0		0.9	2.2
+0.37	7.5		6.2	4.6		5.5	11.2
Mesh							
+3	9.8		8.5	5.6		8.2	16.2
+4	8.9		11.0	6.1		7.3	14.9
+6	9.2		12.2	9.1		6.8	14.7
+8	7.8	0.2	12.5	9.7		6.0	18.7
+10	7.6	0.3	12.2	10.8	9.2	6.2	16.1
+14	7.3	0.5	9.0	9.6	16.1	5.3	0.0
+20	6.2	1.1	5.9	7.2	15.1	4.3	
+28	5.2	2.7	4.0	5.3	12.5	3.8	
+35	4.2	5.6	3.1	4.4	9.6	3.9	
+48	5.8	13.6	4.1	6.8	10.6	7.1	
+65	4.2	12.6	2.6	5.0	6.0	5.8	
+100	2.8	17.2	2.4	4.6	5.7	9.2	
+150	3.1	18.5	1.7	3.0	4.2	6.8	
+200	5.0	13.9	1.0	2.1	2.8	5.6	
−200	5.5	13.8	1.3	3.0	8.2	7.3	
	100.0	100.0	100.0	100.0	100.0	100.0	100.0

* Ore fines, 44.8 pct; flue dust, 20.0; sinter returns, 20.0; roll scale, 4.0; coke breeze, 1.7; water, 9.5 (wet basis). Total carbon, 5 pct.

At this point, the short columns were impregnated once more, using methyl methacrylate monomer. The extremely tough lucite matrix provided by this technique was essential in the preparation of the specimens for final sectioning and polishing. The columns could then be sectioned into slices only 1/16 in. thick, as in sketch 3. After a light grinding, these slices were ready for macrographic study. For work at higher magnifications, 1-in. disks containing the sintering zone were cut from these slices and mounted in conventional briquettes for ease in polishing, sketch 4, Fig. 3.

Raw Materials

Raw materials were used in proportions typical of American sintering practice. Charges were proportioned on a wet basis as follows: Labrador ore fines, 46.5 to 43.3 wt pct; flue dust, 20.0; sinter returns, 20.0; roll scale, 4.0; coke breeze, 0.0 to 3.2; moisture, 9.5. The screen analyses and chemical compositions of each component and of a typical charge are given in Tables I and II.

Microscopic examination of the Labrador ore fines showed that the mineralogical composition varied with particle size. The coarser particles were either limonitic, hematitic, or banded combinations of these minerals. Because the finer limonitic particles coated the coarse pieces, the ore appeared to be entirely limonitic.

Table II. Chemical Analyses of Raw Materials

Constituent	Wt Pct of Constituent					Typical Sinter Mixture,* Dry State
	Ore Fines	Flue Dust	Sinter Returns	Roll Scale	Coke Breeze	
Total Fe	58.11	46.02	63.49	74.53	1.29	55.12
Fe^{++}			47.00	61.30		13.08
SiO_2	7.10	13.11	6.43		4.24	7.25
Al_2O_3	1.24	3.31	1.67		2.48	1.72
CaO	0.07	4.62	1.74		0.32	1.45
MgO	0.08	1.27	0.55		0.13	0.44
P	0.124					0.10
S	0.010				1.15	0.23
Mn	1.03	0.61	0.50			0.77
C		14.40			87.79	5
Volatile matter					1.65	
Ash					9.41	
Combined H_2O						3.85
Combined CO_2						2.60

* Ore fines, 44.8 pct; flue dust, 20.0; sinter returns, 20.0; roll scale, 4.0; coke breeze, 1.7; water, 9.5 (wet basis). Total carbon, 5 pct.

The typical mixture balled readily when moistened and subjected to a mixing and rolling action. A comparison of the size distribution in the dry and in the wet state is shown in Table I. Pelletization occurred by the formation of thick coatings of fine particles around the coarse pieces. Since the major proportion of the fines originated in the ore and in the flue dust, the external color changes in the sintering bed were the result of chemical changes in these fines.

Presentation of Results

Vertical sections of partially sintered beds revealed a number of definite zones or layers in which various physical and chemical changes occurred during the sintering operation. Fig. 4 shows dissection of a partially sintered bed according to the second technique shown in Fig. 2. Although it is not readily discernible in the picture, a very narrow band, called the zone of combustion and sinter formation, is recognized as the region of transition between the coherent sinter cake and the under-

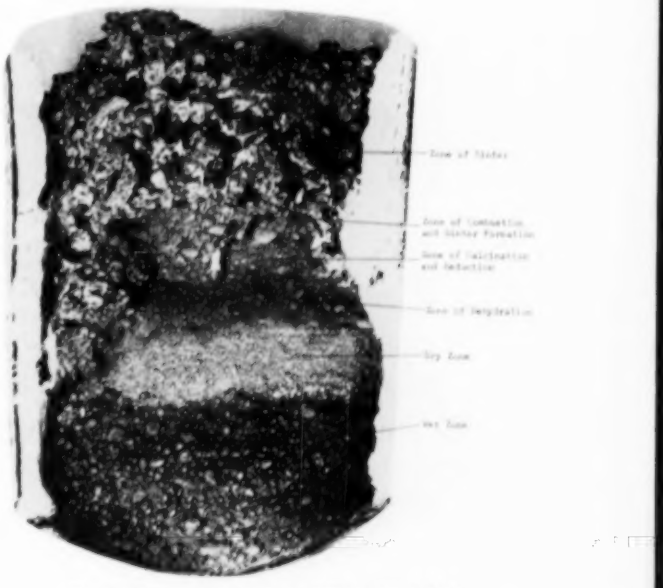


Fig. 4—Zone structure of partially sintered bed.

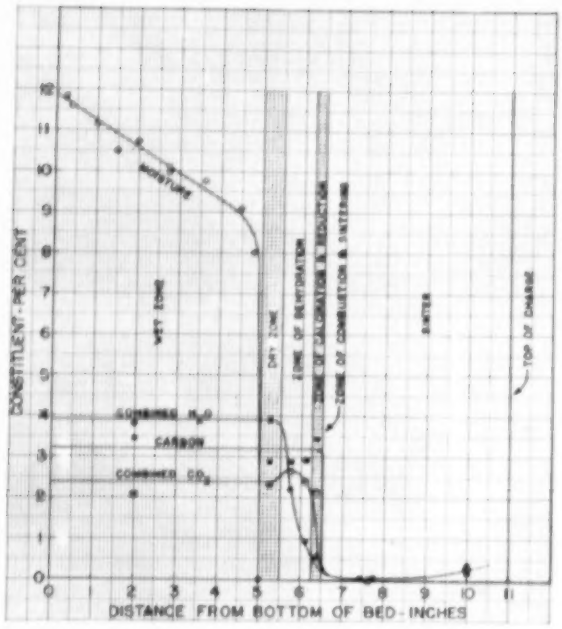


Fig. 5—Variations in chemical analysis in relation to zone structure, low fuel mixture.

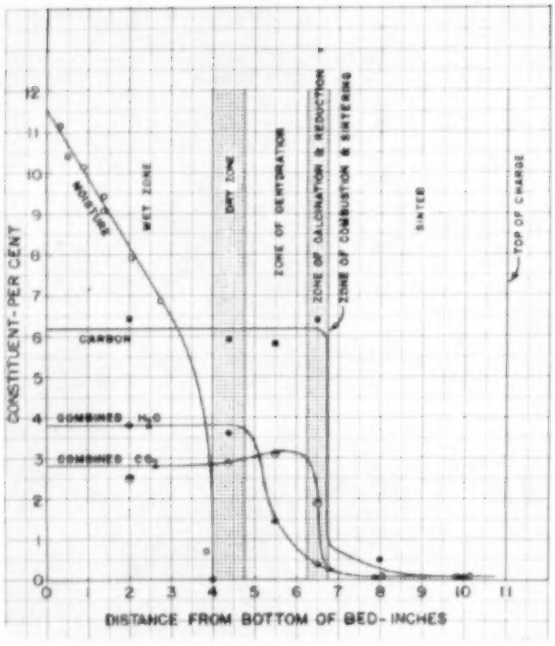


Fig. 7—Variations in chemical analysis in relation to zone structure, high fuel mixture.

lying, granular portion of the bed. The first distinct region of unsintered material is the blue-black zone of calcination and reduction. Directly beneath it is the deep red zone of dehydration. In sharp contrast to this is the light yellow dry zone. Finally, the dark brown wet zone forms the bottom layer of the bed. The white spot at the center of the charge is the thermocouple sheath, and the white areas enclosing the charge are sections of the asbestos lining of the sintering basket.

A series of charges of varying composition, particularly with respect to the fuel, were sintered at various rates of air flow, quenched, and then dissected. Because the zone relationships were qualitatively the same as those just outlined, the pattern was considered to be general. At this point, it seemed necessary to correlate the more important variables of the process with the zone structures.

The most important variables in any sintering operation are those associated with combustion, the quantity of fuel in the mixture, and the rate of air flow through the bed. This investigation was primarily concerned with a study of the effects of variations in the amount of fuel at a constant rate of air flow typical of commercial practice. No direct information has been published on the specific rate of air flow through industrial beds (cubic feet of air per minute per square foot of wind-box area); moreover, estimation is difficult because of the unassessable effect of leakage. The rate of advance of the sintering zone is, however, a direct function of the specific rate of air flow. Expressed in this way, sintering velocities of ½ to ¾ in. per min are typical of plant practices involving 11 to 13-in. beds and suctions of 20 to 25 in. water gage. Throughout the present study, sintering rates of about ½ ipm were obtained with a pressure drop of about 25 in. water gage across the 11-in. bed. It was concluded, therefore, that the specific rate of air flow of 55 cu ft per min per sq ft (stp) was typical of commercial practice. With the air flow held constant at this value, changes in the sintering process were studied as the fuel content was increased from 3 to 6 pct C (dry basis). Using the first method of dissection, Fig. 2, samples were removed for chemical analysis. Plotting the data as a function of zone location gave a *chemical profile* of the partially sintered bed.

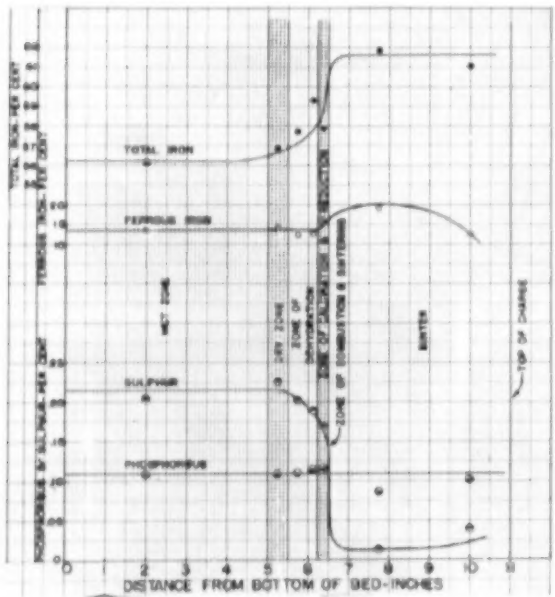


Fig. 6—Variations in chemical analysis in relation to zone structure, low fuel mixture.

Variations in Chemical Analysis Through Partially Sintered Beds

Figs. 5 and 6 summarize the data from the low fuel mixture and Figs. 7 and 8*a* present the findings

of the high fuel charge. Variations in moisture, combined water, combined CO_2, and combustible carbon are shown in Figs. 5 and 7, and changes in total iron, ferrous iron, sulfur, and phosphorus are plotted in Figs. 6 and 8*a*. Zone widths observed from the color changes are also indicated on the graphs.

The Wet Zone—As the hot gases from the combustion zone flow through the unsintered portion of the bed, they are cooled in supporting endothermic reactions and in providing the charge with sensible heat. At the level of the wet zone, the gases are effective only in the removal of moisture from the charge. Figs. 5 and 7 show that the percentage of moisture varied from zero at the junction of the wet and dry zones to almost 12 pct at the grate level for both low carbon and high carbon mixtures. The moisture values are somewhat lower than those existing at the instant combustion was stopped, because of unavoidable losses during quenching and dissection.

Because the sinter mixture contained 9.5 pct moisture, it is apparent that some condensation occurred at the very bottom of the charge. In the 3 pct C charge, the gases became saturated with water after flowing only one fourth of the way through the wet zone. The dew point was not reached in the bed containing 6 pct C, however, until the gas stream had traversed about three fourths of the wet zone. Since the rate of air flow was the same in both cases, this difference was attributed to the much higher temperature of the gas stream in the high fuel charge.

The effects of condensation can be rather serious. If the moisture is high enough to convert the pelletized structure of the feed into a sludge, the permeability of the bed will be sharply decreased. While it is essential to add enough water to pelletize the feed satisfactorily, the moisture content should be minimized to prevent excessive condensation in the wet zone. Sludge formation did not occur in the charges investigated, although aggregates in the bottom layer of the bed were covered with a water film.

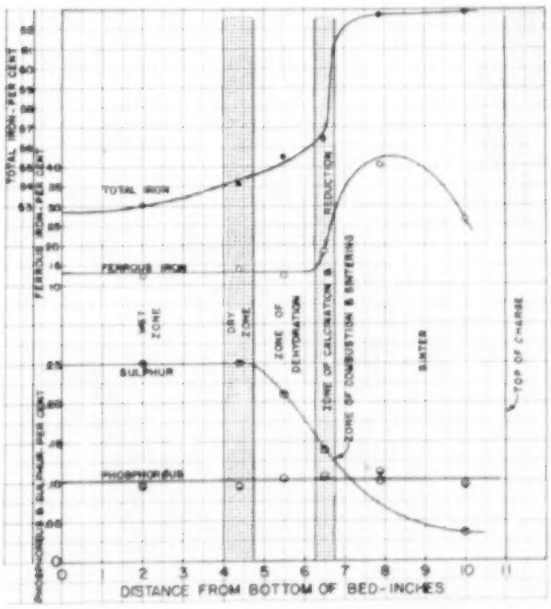


Fig. 8*a*—(Original) Variations in chemical analysis in relation to zone structure, high fuel mixture.

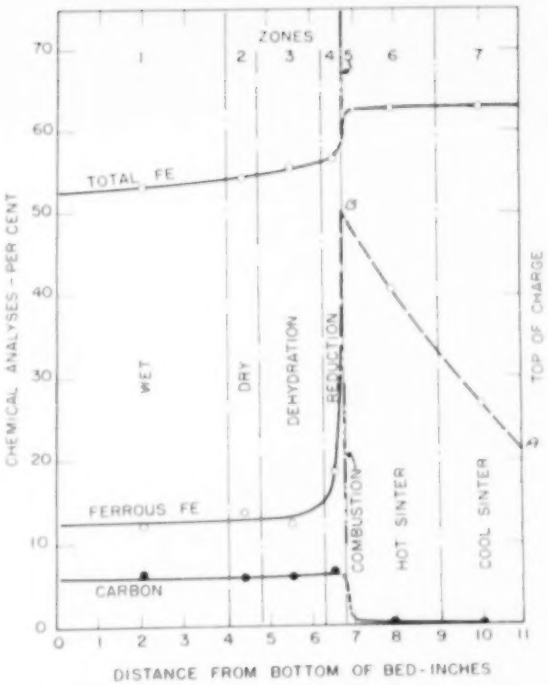


Fig. 8*b*—(Modified) Variations in iron analyses with zone structure, high fuel mixture.

The Dry Zone—As shown by Figs. 5 to 8*a* inclusive, little chemical change takes place within the dry zone, for the materials are not hot enough to initiate the chemical reactions that characterize the overlying zones. This region may be regarded as a zone of preparation, in which the charge is preheated to reaction temperature.

Very little difference was noted between the state of aggregation of the particles in the wet zone and in the dry zone. Heat and chemical effects may cause pellet disintegration above the wet zone. However, the materials used in this study formed tough aggregates, which suffered very little breakdown in the lower zones of the bed. The small amount of dusting observed may have accompanied the evolution of water vapor during drying. During the sintering of some charges, the drop in permeability may be due partly to excessive breakdown of the pellets as they are heated.

Zone of Dehydration—After the charge has been heated to the temperature range of this zone, the hydrated iron oxides begin to dissociate thermally. In both the high fuel and low fuel runs, the elimination of combined water across this zone, Figs. 5 and 7, changes the color from yellow to red.

This zone may also be termed the zone of partial sulfur elimination, for at both fuel levels the sulfur content of this portion of the charge decreased markedly. The removal of sulfur from the fuel, the major source of this element, is usually attributed to simple oxidation. Since hot water vapor is evolved within this layer, some sulfur may be steamed out, as suggested by the work of Powell[7] and Thompson.[8]

The increases in other constituents are probably caused by concentration as a result of the loss of combined water from this zone.

Zone of Calcination and Reduction—At this level of the bed, the gas stream is hot enough to calcine the carbonates, and it also contains a sufficient con-

centration of carbon monoxide to reduce a part of the hematite to magnetite. The curves for combined CO_2 in Figs. 5 and 7 indicate that decomposition of the carbonates is practically complete within this zone for both high fuel and low fuel mixtures. Reduction of hematite to magnetite is shown by the increase in ferrous iron across this zone, Figs. 6 and 8*a*. Further evidence of reduction at this level is the familiar change in color from red to blue-black. Moreover, the particles of this zone were all strongly magnetic. The degree of conversion to magnetite was greater for the high fuel charge than it was for the low fuel mixture because of the higher concentration of reducing agents. Further changes within this region are the continued removal of sulfur and the sharp increase in total iron by concentration. In agreement with other studies,[5] the elimination of sulfur is favored by lower percentages of carbon in the mixture.

It was not until the arrival of the zone of calcination and reduction that the aggregates of particles began to break down. The coatings of fine particles that were built up around the coarse pieces during mixing were observed to spall away from their central kernels and to disintegrate. Most of the coarse pieces were badly cracked. The extent of breakdown increased rapidly as the particles were heated to peak temperature. Actually, at the junction of this zone with the overlying sinter cake, the material was little more than an incoherent powder in many areas.

Zone of Combustion and Sinter Formation— Figs. 5 and 7 show that the percentage of carbon remains fairly constant throughout all the zones of unsintered material. The abrupt decrease from the amount in the charge to a very low value marks the zone of combustion. Within a very narrow region, a large quantity of heat is released, and much of it is absorbed by the small, contiguous particles. The particles then fuse, flux, or dissolve in the adjacent viscous mass. Therefore, this region is also referred to as the zone of sinter formation. The hot products of combustion are drawn out of this zone of maximum temperature and alter the lower, unsintered portion of the bed in accordance with the trends.

Table III. Zone Dimensions

Region	Zone Width, Inches	
	Low Fuel Mixture, 3 Pct C	High Fuel Mixture, 6 Pct C
Wet zone material above the dew point level	1.25	2.75
Dry zone	0.50	0.75
Zone of dehydration	0.75	1.50
Zone of calcination and reduction	0.25	0.50
Total of all intermediate zones	2.75	5.50

The sulfur is practically burned out at this point in the low carbon charge, but a substantial amount still remains in the high fuel mixture. Further reduction of hematite raises the ferrous iron to the maximum percentages found within the sinter cake, and increases the total iron by concentration.

Zone of Sinter—The zone of sinter is a broad zone of the finished product. Chemical change within the sinter seems to be restricted to a partial reoxidation of the magnetite by the stream of air in the early stages of cooling. In the high fuel charge, the ferrous iron was decreased from a maximum of over 40 pct to about 26 pct by reoxidation of the upper part of the sinter, Fig. 8*a*. Less reduction occurred in the low carbon charge and reoxidation decreased the ferrous iron to a value no greater than that of the original mixture, Fig. 6. Visual evidence of reoxidation was found in the more reddish cast of the upper, cooler sinter, and in the abundance of microscopic crystals of hematite around the edges of the pores in this region. The sinter closer to the combustion zone was blacker and contained few grains of hematite.

In the high carbon mixture, the final sulfur content was somewhat more than twice that of the residual sulfur in the low fuel charge. Considering the overall reduction in weight during sintering, the horizontal phosphorus curves indicate that only a very small amount, if any, of this element was lost in the operation.

A substantial difference was found in the strengths of the top and bottom halves of the sinter cake. The upper half was noticeably weaker than the lower portion, and the difference was greatest for the low carbon mixture. This weakness may be due to the relatively poor preheat and low peak temperatures attained in the top portion of the charge.

Zone Dimensions of the High Fuel and Low Fuel Mixtures—Because the specific rate of air flow was the same for both cases, the peak temperature in the combustion zone was considerably higher in the high fuel mixture than in the low carbon charge. The widths of the intermediate zones, therefore, were greater in the former case than in the latter.

The term *intermediate zones* refers to all portions of the bed between the sinter cake and the dew point level of the wet zone. As the width of the sinter cake increased, the width of the wet zone decreased by about the same amount. The dimensions of the *intermediate zones*, however, remained fairly constant through most of the run. The widths of these zones, tabulated for easy comparison, are shown in Table III.

The intermediate zones constitute the entire region of transition from the initial moist charge of 9.5 pct water to the final sinter cake. Although their total breadth was twice as great in the 6 pct C mixture as it was in the 3 pct C charge, the change from the wet charge to the immature sinter occurs over a surprisingly short distance.

Sintering Zone—The thin slices sectioned from the lucite-impregnated bed, as in sketch 3 of Fig. 3, were suitable for photographing the sintering zone. Macrographs were obtained from several specimens and a composite print was made, which reproduced a wide section of the bed at actual size, Fig. 9. The granular, unsintered portion of the bed is in sharp

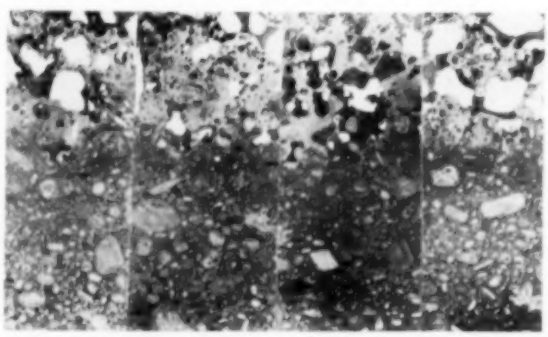
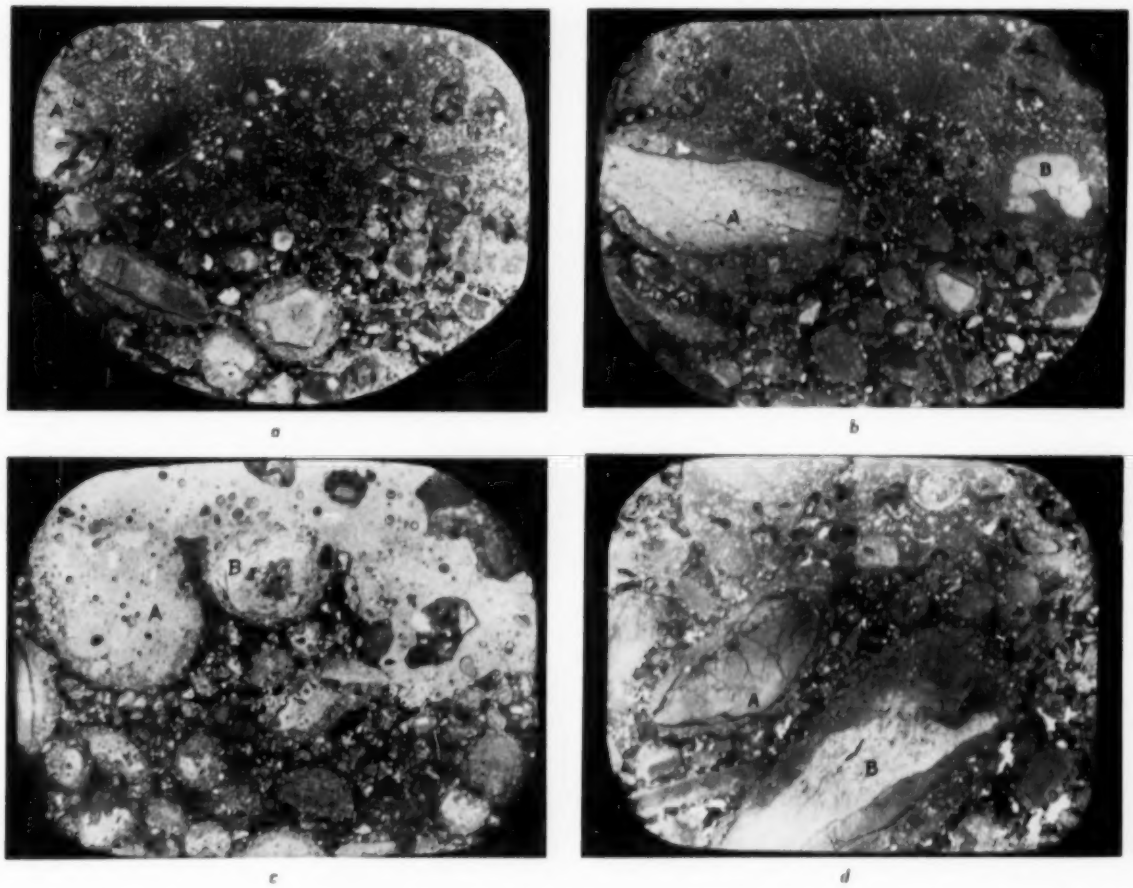


Fig. 9—Micrograph of zone of combustion and sinter formation. Area reduced approximately 50 pct for reproduction.



a *b* *c* *d*

Fig. 10—Micrographs of zone of combustion and sinter formation. *a*, *b*, and *d*, *A* and *B*, ore; *c*, *A*, return sinter, *B*, ore. X10. Area reduced approximately 50 pct for reproduction.

contrast to the vitreous, cellular appearance of the sinter cake. The sintering zone is, however, rather irregular and diffuse. This lack of definition is unavoidably associated with the raw materials used. Chemically, the system was highly heterogeneous, and the various components were all present in a wide range of particle sizes. As a result, local differences in the conditions of heat transfer and in the fusibility of the materials produced a broad, undulating zone of sintering. It was impossible to locate a clearly representative sintering *front* in a highly magnified field. Typical views of the sintering zone are shown in Figs. 10*a*, *b*, *c*, and *d*.

Fig. 10*a* shows the sintering front advancing upon a cluster of particles ranging in size from fines to aggregates of medium size. The sinter at the top is easily recognized by its gray color and impervious structure. The tendency of the fines to form shells around the coarse pieces has produced pellets whose diameters are about twice those of the kernels. The plastic sinter has engulfed a large piece of ore, *A*, and solution is progressing at the edges of the particle. Internal cracking is evident.

Fig. 10*b* shows two large pieces of ore in varying stages of ingestion by the igneous solution. The larger piece of ore, *A*, has just been contacted by the fused mass and, because of its size, it is probable that complete solution would not have occurred had sintering continued. An outer shell of fines is clearly discernible. The smaller piece of ore, *B*, has been enveloped by the liquid phase. Reduction to magnetite seems to have occurred over its entire surface before the sintering zone arrived. The white center is hematite; the gray peripheral zone, magnetite.

In Fig. 10*c*, a large piece of return sinter, *A*, and an adjacent fragment of ore, *B*, are being gradually absorbed by the plastic or semimolten mass. The coating of fines on the lump of return sinter is the only means of distinguishing it from the new sinter. The ore particle suffered extensive cracking during heating and partial reduction.

A wide variety of particle sizes are shown in Fig. 10*d*, including two large pieces of ore that dominate the field. Reduction of the smaller piece of ore, *A*, almost entirely to gray magnetite was probably accelerated by the extensive cracking. In the larger ore lump, *B*, the progress of reduction is revealed by the distinct layer of gray magnetite around the core of white hematite. The difficulty of converting large pieces of ore into a coherent mass of sinter is clearly evident from these photographs.

Ferrous Iron Gradient in the Iron Ore Sintering Process

The variations noted in iron analyses across the partially sintered charges were significant and deserve further comment, particularly in the high fuel mixture. In this respect, Fig. 8*b* reveals that the sinter zone was running as high as 40 pct ferrous iron and 62 pct total iron. A simple stoichiometric calculation shows that these analytical results would

Table IV. X-Ray Examination of Reaction Zones in a Partially Sintered Iron Ore Bed. High Fuel Mixture: Iron Radiation

Zone No.	Zone Designation	Total Iron Analysis, Average Pct	Ferrous Iron Analysis, Average Pct	X-Ray Results: Phase Content, Pct	X-Ray Results: Lines Found
3	Dehydration	55.52	12.19	60 hematite 40 magnetite Trace foreign	
4	Reduction	56.40	18.48	75 magnetite 25 hematite Trace foreign	
6	Hot sinter	62.76	40.39	60 magnetite 40 wüstite Trace foreign	d_n values checked for 8 wüstite and 14 magnetite lines.
7	Cool sinter	62.85	26.53	95 magnetite Trace hematite Trace wüstite	

require 45 pct of the iron to be in the wüstite state with the remainder as magnetite. Such a high wüstite content was unexpected and indicated that reducing conditions may be quite high within iron ore sintering beds, and that the wüstite phase may be important in the overall mechanisms of the process.

Since the results of chemical analyses in themselves are not always reliable in predicting phase relationships, the presence of the wüstite phase was recently checked by means of the X-ray diffraction method. In these experiments portions of the same samples as had been used for the chemical analyses were X-rayed by the powder method and the resulting patterns compared with standard iron oxide patterns for positive identification. The pattern from the hot sinter zone was also carefully measured, and the resulting d_n values for the various lines were checked against the standard data as given in the literature. The final results for the dehydration, reduction, and sinter zones are summarized in Table IV, along with an estimate of their phase content from visual comparison with phase standards. The high percentage of wüstite phase in the hot sinter zone is definitely confirmed and the estimated amount checks well with the results calculated from the chemical analyses.

Inspection of the original figure, Fig. 8*a*, reveals that the ferrous iron relationship could have been more judiciously drawn as shown in the modified figure, Fig. 8*b*, accompanying this discussion. A number of reasons support such a change. For instance, most high carbon charges, when sintered to completion, yield sintered products composed mainly of magnetite, and thus running about 20 pct ferrous iron. This value of ferrous iron could then well serve as an anchoring point, *A*, at the extreme top of the charge in plotting the overall ferrous iron gradient. Furthermore, the two ferrous iron points shown in the sinter layer of the original, Fig. 8*a*, represent the *average composition* of the top and bottom halves of the sinter zone, respectively. The original plot thus shows the average point in the bottom half of the sinter zone at 40.19 pct ferrous iron, and as a maximum in the ferrous iron relationship. Obviously, this condition is unrealistic. Points of higher ferrous iron content must exist in this layer, and it is logical to assume that they are located at the zone of combustion and sintering where the strongest reducing conditions are generated. All of these ideas have been used in redrawing the ferrous iron plot as shown in Fig. 8*b*.

The sketch as now shown indicates that extremely high ferrous iron contents may exist at the zone of combustion and sintering, *B*—possibly running over 50 pct ferrous iron. More realistically, the phase relationships under these conditions would dictate that over 70 pct of the iron be present in the form of the wüstite phase. These conclusions strengthen the contention that the wüstite phase plays an important role in the sintering of high fuel charges. The mechanisms must involve a reduction of much of the iron to the ferrous state at the sintering interface, with a consequent reoxidation of the sintered material by the incoming air stream.

Summary

Dissection procedures were developed for immediate and accurate sampling of all the zones in partially sintered beds. Six zones were clearly marked by changes in color. From the top to the bottom of the bed, these zones included: 1) the cake of sinter, 2) the narrow zone of combustion and sinter formation, 3) the zone of calcination and reduction, 4) the zone of dehydration, 5) the dry zone, and 6) the wet zone. With 3 pct and 6 pct C in the mixture, dry granular materials were converted into a coherent mass of sinter within distances of 1.5 and 2.75 in., respectively.

The chemical composition changed gradually across the various zones. The ferrous iron increased in the zone of calcination and reduction and attained a maximum in the freshly formed sinter. Further out in the sinter cake, the percentage of ferrous iron decreased as a result of air oxidation. The extent of increase in ferrous iron and the final amount in the sinter appears to depend upon the amount of fuel used. A substantial amount of sulfur disappeared from the zone of dehydration, presumably by reaction of steam with the coke. In accordance with general practice, sulfur was eliminated more completely in the low fuel charge.

Methods were developed for impregnating and sectioning partially sintered beds of iron ore. This permitted a study of the very narrow sintering zone, which advances as portions of the charge soften, fuse, and merge into the cake of sinter. Typical vertical sections extending across the sintering zone are included to show how the zone advances. Gross discontinuities were observed in the relatively coarse mixture investigated. Charges of finer materials should be studied to relate changes across the sintering zone with changes in raw materials, and in important variables of operation.

Acknowledgments

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Vernon Bye, of the Minnesota Mines Experiment Station, is credited with a portion of the analytical work.

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Technical Note

Metallographic Identification and Crystal Symmetry of Titanium Hydride

by L. D. Jaffe

IN previous metallographic work on titanium and its alloys, difficulty has been encountered in distinguishing spheroidal particles of titanium hydride, dispersed in α-titanium, from other phases that may be present, such as β-titanium. This problem is common in examination of commercial unalloyed titanium.[1]

In the course of an investigation of the Ti-H phase diagram, it was noted that titanium hydride, dispersed in a matrix of α-titanium, showed strong optical anisotropy when examined under polarized light. With sensitive tint illumination, particles of the hydride, when the stage is rotated, changed from bright yellow to bright blue in color. Since β-titanium is cubic, it showed no color change on rotation, remaining dark gray or blue-gray. Although α-titanium is hexagonal, it showed only a minor color change, going from pale pinkish-blue to pale purplish-blue. These colors could be modified somewhat by adjustment of the illumination. They were observed in specimens mechanically polished, both without etching and after etching with hydrofluoric acid-nitric acid mixtures in either water or glycerine. The strong color change of the hydride seems to offer a simple method for its metallographic identification.

Optical anisotropy of titanium hydride is inconsistent with its accepted face-centered-cubic structure.[2] Samples of iodide titanium containing 6 to 40 atomic pct deuterium were prepared for diffraction studies. After deuterating and slow cooling[3] the samples were held 21½ hr at 400°C, water quenched, held 61 hr at 100°C and quenched, then held 64½ hr at 255°C and quenched. Debye exposures with Mo$K\alpha$ X-rays and 1.111A neutrons were made at room temperature of samples as-deuterated and after the 100° and 255°C treatments. Table I shows the lines observed, in addition to those attributable to α-titanium. These data show that the hydride phase was not cubic, but probably tetragonal. The titanium atoms appeared to form a body-centered-tetragonal lattice complex, two titanium atoms per cell, with approximate dimensions:

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Table I. Diffraction Data

Interplanar Spacing, A	Relative Integrated Intensity		Lattice Complex Indexing	
	X-Ray*	Neutron*	BCT†	FCT†
2.98	0	25	‡	(101)‡
2.50	50	<35	(101)	(111)
2.20	<100	<100	(110)	(200)
2.10	10 (?)	50	(002)	(002)
1.81	0	20	‡	(211)‡
1.55	30	30	(200)	(220)
1.33	<35	<50	(211)	(311)
1.27	<7	<100	(103)	(113)
1.01	10		(301)	(331)
0.99	0	<25	(104)‡	(114)‡
			(310)	(420)
			(213)	(313)

* < indicates lines overlapped by lines of α-titanium, so only maximum limit to intensity of hydride line could be set.
† BCT and FCT indicate body-centered-tetragonal and face-centered-tetragonal, respectively.
‡ ‡ indicates neutron lines not fitting indexing of lattice complex of titanium atoms, and presumably arising from scattering by deuterium atoms.

$a = 3.12$A, $c = 4.18$A, and $c/a = 1.34$A. This is equivalent to a face-centered-tetragonal complex, four titanium atoms per cell, with approximate dimensions: $a = 4.42$A, $c = 4.18$A, and $c/a = 0.946$A. The presence of the 2.98 and 1.81A neutron lines indicates that the true Bravais lattice is not identical with this lattice complex, but may be primitive tetragonal with dimensions some multiple of those of the complex.

The relative intensities of the hydride and α-titanium X-ray lines as functions of overall composition indicated that the hydride, presumably in equilibrium with the α at 100° to 255°C, contained more than 62 atomic pct H.

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Equilibria of Sulfur and Oxygen Between Liquid Iron and Open Hearth-Type Slags

Data of several studies on the equilibrium between molten iron and open hearth-type slags have been combined to determine some of the chemical reactions involved in steelmaking. Effects of slag composition and temperature on the iron oxide activity, distribution of sulfur between slag and metal, and the carbon content of the metal are discussed. Also, an overall reaction for the equilibrium between sulfur and oxygen is presented.

by H. L. Bishop, Jr., H. N. Lander, N. J. Grant, and J. Chipman

DURING the past 15 years, a number of significant equilibrium studies have been carried out between liquid iron and simple basic and acid slags in an effort to throw light on the chemical reactions involved in steelmaking. This paper is a summary of the experimental reports on investigations which have been conducted at the Massachusetts Institute of Technology, and of related work in other laboratories. The investigations summarized in this paper include the works of Fetters and Chipman[1] and Taylor and Chipman,[2] which dealt with simple slags that are similar to those found in the open hearth. The sulfur determinations of the slag-metal tests of Fetters and Chipman as reported by Grant and Chipman[3] are included. Also, the investigations by Winkler and Chipman[4] and Grant and Chipman[5] of more complex slag systems are included in this summary. The experimental procedure is described in detail in each of these papers and will not be discussed.

Since the previous reports have been somewhat independent, it is desirable to review and combine the data of the various investigations to determine, as accurately as possible, the reactions in the slag-metal systems of the open hearth. It is the object of this report to present the combined data in such a way as to enhance their utility and applicability.

Oxidizing Power of Slags of the System (CaO+MgO)-SiO_2-FeO in Equilibrium with Carbon-Free Iron

The slags selected to determine the iron oxide activity in equilibrium with pure liquid iron were those containing less than 2 pct each of P_2O_5, MnO, or Al_2O_3 on a weight percentage basis. The method used to calculate activities of iron oxide in liquid slags was that developed by Taylor and Chipman.[2] The solubility of oxygen in molten carbon-free iron under pure iron oxide slags can be expressed by the following equation

$$\log (\text{pct O}) = \frac{-6320}{T} + 2.734 \qquad [1]$$

where T is the absolute temperature. The oxygen solubility values as determined from Eq. 1 were used throughout this study to calculate the iron oxide activity.

The iron oxide activities of slags of various iron oxide contents are plotted in Fig. 1 as a function of the basicity, which is defined as the ratio (pct CaO + pct MgO)/pct SiO_2, and are based on molecular percentage values of the components. The percent $(FeO)_t$ is the total iron content in the slag calculated to FeO. Fig. 1 shows that as the basicity increases at constant mol pct $(FeO)_t$, the iron oxide activity increases rapidly and reaches a plateau which extends on the basicity scale from about 1.3 to 2.3; the iron oxide activity then decreases rapidly until a basicity of about 3.0 is reached, where it tends to level off.

The solid portions of the curves represent the experimental data of Fetters and Chipman and Winkler and Chipman. Their experimental heats were made in magnesia crucibles, and hence the slags were saturated with magnesia. The broken lines on the left represent the experimental data of Taylor and Chipman, who used a rotating crucible. In general, these slags contained less than 3 pct MgO.

The absence of data for slags containing less than about 40 mol pct $(FeO)_t$ and basicity ratios greater than about 2.5 made it necessary to calculate curves using ionic concepts of slags. The broken lines on the right, which disregard the solubility limits of di and tricalcium silicate and lime, were obtained by following the treatment of Flood and Grjotheim,[6]

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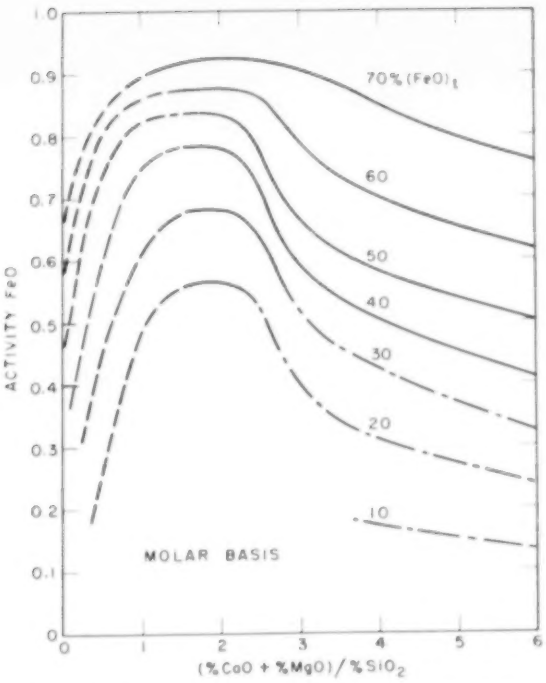

Fig. 1—Iron oxide activity vs molar basicity ratio at various iron oxide concentrations: system (CaO + MgO) — SiO_2 — $(FeO)_t$.

and were adjusted to agree with the experimental data at 40 mol pct $(FeO)_t$.

The curve for 10 mol pct $(FeO)_t$ terminates at a basicity ratio of about 3.5 because of lack of data at lower values. At low basicity ratios, the curves for 20, 30, and 40 mol pct $(FeO)_t$ terminate at the composition corresponding to silica saturation at 1600°C, as obtained from the CaO-SiO_2-$(FeO)_t$ phase diagram. The curves representing the iron oxide activities at 50, 60, and 70 mol pct $(FeO)_t$ extend on the basicity scale to zero, which corresponds to the $(FeO)_t$-SiO_2 binary system.

Comparison with Other Results—The iron oxide activity of the slags corresponding to a basicity of zero can be compared with the results of Schuhmann and Ensio,[5] Michal and Schuhmann,[7] and Rey.[8] Schuhmann and Ensio determined the iron oxide activity by equilibrating CO_2-CO gas mixtures with iron-silicate slags in iron crucibles at temperatures from 1250° to 1400°C. Within the limits of experimental error, they found that the iron oxide activity did not vary for a given silica concentration over the temperature range studied. From these activity-temperature data, they estimated that the partial molal heat content of iron oxide in the slag, relative to a pure iron oxide melt in equilibrium with γ-iron, was zero.

The partial molal heat content of iron oxide can be evaluated also from the heat of mixing diagram for the iron oxide-silica system as presented by Rey. Curve No. 2 in Rey's Fig. 1 is in complete agreement with the results of Schuhmann and Ensio for slags less acid than the orthosilicate; that is, the partial molal heat content of iron oxide is zero. Since 70 mol pct $(FeO)_t$ is less acid than the orthosilicate composition, the value of 0.65 for the iron oxide activity as found by Schuhmann and Ensio is applicable for the case of liquid iron, and is in complete agreement with Fig. 1.

Values of the iron oxide activity of slags more acid than the orthosilicate composition in equilibrium with molten iron at 1600°C were calculated by using the low temperature data of Schuhmann and associates and the partial molal heat content of iron oxide as presented by Rey together with the following equation

$$\overline{L}_{FeO} = \text{approximately } R\left(\frac{\partial \ln a_{FeO}}{\partial\left(\frac{1}{T}\right)}\right)_{N_{SiO_2},\, a_{Fe}=1}$$

where $\overline{L}_{FeO}$ is the partial molal heat content of iron oxide, R is the gas constant, T is the absolute temperature, and a_{FeO} is the iron oxide activity. This calculation gives values of 0.40 and 0.53 for iron oxide activities at 50 and 60 mol pct $(FeO)_t$, respectively. These values are in close agreement with results shown in Fig. 1, which give values of 0.45 and 0.57 at 50 and 60 mol pct $(FeO)_t$.

Elliott's study of existing slag-metal equilibria data[9] is in close agreement with the results shown in Fig. 1. In this study, the data of Taylor and Chipman and Schuhmann and Ensio, and the slags of Winkler and Chipman, which contained low MnO and P_2O_5 concentrations, were used to establish the positions of their curves plotted in a ternary composition diagram.

The results of a review of available slag-metal equilibria data by Turkdogan and Pearson[10] are not in complete agreement with the results shown in Fig. 1. As shown by their method of plotting the data, the maximum in the iron oxide activity curve shifts to higher basicity values as the iron oxide concentration increases. This tendency, which does not exist in the present study, may be due in part to the use of data dealing with complex slags containing appreciable quantities of MnO and P_2O_5.

With regard to P_2O_5, Turkdogan and Pearson indicate that it is equivalent to SiO_2 in its effect on the iron oxide activity for concentrations up to 0.10 mol

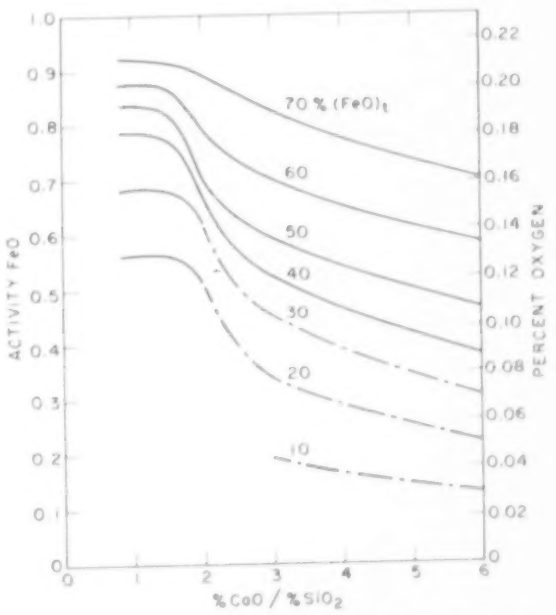

Fig. 2—Oxygen in metal at 1600°C (and iron oxide activity) vs the molar lime-silica ratio in magnesia saturated slags at various molar iron oxide concentrations.

fraction. The study by Grant and Chipman on sulfur equilibria between liquid iron and slags, and further work reported in this paper on sulfur equilibria, suggest that P_2O_5 should be calculated as $PO_{2.5}$. Some of the difference between the two studies may be due also to the use by Turkdogan and Pearson of Korber and Oelsen's[11] slag-metal equilibria data with Taylor's FeO solubility data. More consistent results would be expected if Korber and Oelsen's data were used throughout.

Effect of Temperature—Activities of iron oxide, plotted as a function of the basicity as presented in Fig. 1, are applicable for any temperature and slag

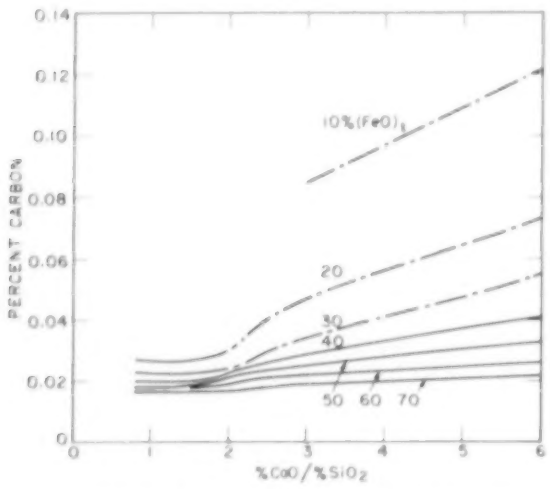

Fig. 3—Minimum carbon content of metal at 1600°C vs the molar lime-silica ratio in magnesia saturated slags at various molar iron oxide concentrations.

except for slags of low basicity ratios having iron oxide concentrations less than about 67 mol pct. Fetters and Chipman have shown that within the limits of experimental error the temperature dependence of the oxygen solubility is the same for all basic slags; hence, for all basic slags of given compositions, the iron oxide activity is the same regardless of temperature. The iron oxide activities of slags having both a low basicity value and an iron oxide concentration less than 67 mol pct are probably temperature dependent, as mentioned previously.

Iron Oxide Activity and the Oxygen Concentration as a Function of the Molar Lime-Silica Ratio—The oxygen content of the metal at 1600°C and the iron oxide activity are presented in Fig. 2 as a function of the molar lime-silica ratio, hereafter referred to as the *V*-ratio. The data used were obtained from studies involving simple magnesia saturated slags,[1, 4] and the calculations were based on the method of Taylor and Chipman.[3] The lack of data for magnesia saturated slags having *V*-ratios of less than about 0.8 made it necessary to terminate the lines as indicated in Fig. 2. These curves can be used to represent the equilibrium conditions in the basic open hearth furnace when the slags contain small percentages of P_2O_5, Al_2O_3, and MnO.

In open hearth practice, the oxygen content of metal in contact with a particular slag will hardly ever be as high as the values indicated in Fig. 2 because of the carbon boil. These values, however, can be used to determine the oxygen potential existing between the slag and metal, which in turn will affect the rate of oxygen transfer.

In Fig. 3, the limiting carbon contents of metal at 1600°C were calculated from the oxygen levels given in Fig. 2 and are plotted as a function of the *V*-ratio at various iron oxide concentrations. These curves represent the concentration of carbon in the metal in equilibrium with magnesia saturated slags and a CO pressure of 1.5 atm. The activity coefficients of oxygen and carbon have been taken from a summary in *Basic Open Hearth Steelmaking*[12] and the carbon monoxide is assumed to be nucleated at a pressure of 1.5 atm in accordance with the conclusions of Murphy and Miltenberger.[13] Actually, the carbon content of open hearth heats will almost always be greater than these values.

A point of practical interest, as shown in Figs. 2 and 3, is that the oxidizing power of the slag is determined not only by the iron oxide content, but also by the *V*-ratio. A rapid rate of oxygen transfer to the metal should occur, with a minimum iron loss to the slag, when operating with slags having *V*-ratios of about 0.8 to 1.8 and low iron oxide concentrations.

Solubility of Magnesia in Slag

The data of Fetters and Chipman[1] were used to determine the solubility of magnesia in slags of the system CaO-MgO-SiO_2-FeO. They noted that the magnesia was not in suspension but was in solution in the slag. In nearly pure iron oxide slags their data were represented by the equation

$$\log (\text{wt pct MgO}) = \frac{-6600}{T} + 4.02.$$

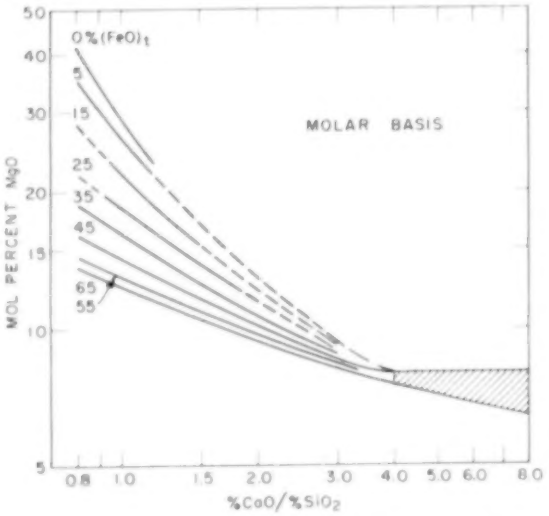

Fig. 4—Solubility of magnesia as a function of the molar lime-silica ratio at various iron oxide concentrations.

Slag samples which were subjected to careful petrographic examination did not contain any periclase. It was noted that after the heat had been molten for 1 or 2 hr, the magnesia content did not increase further except as a result of changes in the temperature or slag composition.

Changes in the solubility of magnesia with slag composition are shown in Fig. 4. Here the mol pct MgO is plotted against the *V*-ratio at various iron oxide concentrations. It is to be noted that the solubility of magnesia decreases as the iron oxide content increases; at a constant iron oxide concentration, the solubility decreases as the *V*-ratio increases.

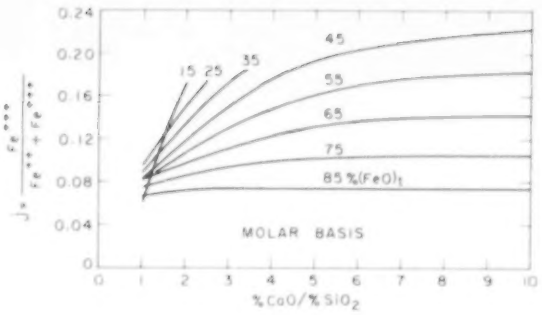

Fig. 5—The molar ratio of ferric iron to total iron (*j*-ratio) in magnesia saturated slags as a function of the molar lime-silica ratio at various iron oxide concentrations.

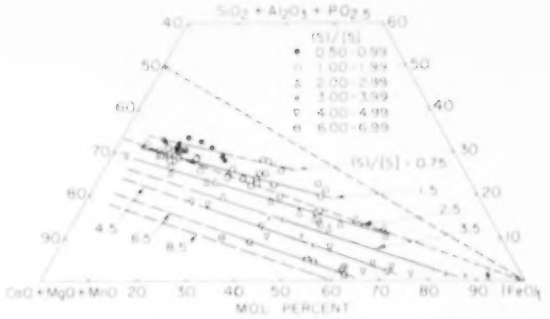

Fig. 6—Desulfurization ratio for the adjusted ternary slag system CaO-SiO$_2$-(FeO)$_t$ in the temperature range 1570° to 1630°C.

The solid portions of the curves in Fig. 4 represent experimental magnesia solubility values, with the exception of the curve at 0 pct iron oxide. The curve representing the solubility of magnesia at 0 mol pct (FeO)$_t$ is based on the MgO-CaO-SiO$_2$ phase diagram at 1600°C. The two sources of magnesia saturation values are in good agreement. The broken curves have been extended essentially parallel to the solid lines but it is not inferred that slags of these compositions exist at 1600°C. The extended lines may have application if the range of fluid compositions is broadened by the addition of other oxides such as MnO, Al_2O_3, and P_2O_5. The shaded area represents the scatter of points for slags containing basicity ratios above 4.0.

The data used to determine the solid portions of the curves varied in temperature from 1540° to 1750°C and were not corrected to any one temperature. A careful study of the data indicates that the temperature effect is small, and since most of the heats were equilibrated at temperatures from 1550° to 1650°C, it is reasonable to assume that the curves represent the solubility of magnesia at about 1600°C. The variation of the magnesia concentration was seldom greater than about ±2 pct for the 5 to 45 mol pct and ±1 pct for the 55 to 65 mol pct iron oxide concentrations. These facts strengthen the validity of the assumption that the curves represent magnesia saturation values at about 1600°C.

Ferric Iron in the Slag

In a study by Larson and Chipman,[4] the ratio of ferric iron to total iron of iron oxide slags, hereafter referred to as the *j*-ratio, was established by equilibrating the slag with air or mixtures of CO_2 and CO. They were able to establish values of the *j*-ratio over a range of oxygen pressures of 10^{-1} to 10^{-8} atm, using their results and existing data on slag-metal equilibria.

They found that the addition of CaO and BaO to iron oxide increased the *j*-ratio considerably, but that smaller increases occurred with MnO and MgO additions. It was also found that the *j*-ratio decreased with additions of acid oxides such as SiO_2 and TiO_2, and was dependent on the ratio of acid to base when additions of slag containing CaO and SiO_2 were made. Additions of slag containing a molar ratio of lime to silica of 2.24 increased the *j*-ratio. With slag having a lime-silica ratio of 1.31, additions up to 60 pct CaO+SiO_2 increased the *j*-ratio. Beyond 60 pct a decrease resulted. All additions of slag having a molar ratio of 0.54 also decreased the *j*-ratio.

St. Pierre[5] has also made a thorough study of the values of the *j*-ratio of iron oxide slags at 1550°C in equilibrium with an atmosphere obtained by reacting a gas mixture having a ratio of SO_2/CO = 34.1, which corresponds to oxygen and sulfur pressures of 1.03×10^{-8} and 6.20×10^{-4} atm, respectively. The effect of the additions of slag containing CaO and SiO_2 on the *j*-ratio was dependent upon the base to acid ratio in much the same manner as the slags studied by Larson.

The data of Fetters and Chipman[1] were used to calculate the ratio of the ferric iron to total iron on a mol fraction basis. This ratio is plotted against the *V*-ratio at various iron oxide concentrations in Fig. 5. The *j*-ratio increases as the *V*-ratio increases. However, the rate of increase is dependent upon the iron oxide concentration of the slag.

The curves of Fig. 5 represent the data for magnesia saturated slags in equilibrium with liquid iron over a temperature range of 1540° to 1750°C. The temperature effect was slight but it was noticed that the higher temperatures resulted in lower *j*-ratios. The data were not corrected to any one temperature, but the curves in Fig. 5 probably best represent the condition at 1600°C. In general, the variation in the *j*-ratio over a 10 pct range in iron oxide content did not exceed ±0.02 *j*-units.

Fig. 5 shows that the addition of a slag containing lime and silica with a *V*-ratio greater than about 1.6 to iron oxide increases the *j*-ratio. The addition of small amounts of a less basic slag increases the *j*-ratio at first, but larger additions decrease the *j*-ratio. In general, these trends are in agreement with the results of Larson and Chipman and St. Pierre as established through gas-slag equilibria.

Sulfur Equilibria Between Slag and Metal

Recent investigations have shown the quantitative relationships of slag-gas and the slag-metal equilibria which are important to sulfur control. The following is a summary of the equilibria of sulfur between slag and carbon-free iron.

Distribution of Sulfur Between Slag and Metal—A measure of the desulfurizing power of a slag in equilibrium with metal is the distribution ratio defined as the ratio of the weight percentage of sulfur in the slag to the weight percentage of sulfur in the metal, (pct S)/[pct S]. The experimental data selected to study the distribution of sulfur are those of Fetters and Chipman,[1] Winkler and Chipman,[2] and Grant and Chipman.[3]

In Fig. 6, the sulfur distribution ratio is plotted against the slag composition in the quasi-ternary

diagram $(CaO + MgO + MnO)$-$(SiO_2 + Al_2O_3 + PO_{2.5})$-$(FeO)$, on a mol fraction basis. In order to eliminate the effect due to temperature extremes, only the samples taken in the interval 1570° to 1630°C were used to establish the curves. The basic constituents were considered to be of equal strength on a molar basis in their effect on the sulfur distribution ratio. The P_2O_5 was calculated as $PO_{2.5}$, which was treated as equivalent to Al_2O_3 and SiO_2. The mol fraction of iron oxide was calculated by converting the total mols of iron to FeO, and is represented by the symbol $(FeO)_t$. This study excluded slag tests containing CaF_2, but did include slags containing up to 15, 17, and 35 mol pct Al_2O_3, $PO_{2.5}$, and MnO, respectively. The generally good correlation substantiates the utility of this method of calculating the slag constituents.

Again, in the construction of Fig. 6, the solubility limits of di and tricalcium silicate and lime at 1600°C for the simple ternary system were assumed to be suppressed by the presence of other oxides, such as MnO, Al_2O_3, and P_2O_5. Hence, the linear relationship existing in the liquid phase field of the complex slag system extends over a greater range than in the case of the simple ternary system. The solid lines represent the range of composition over which data are available. Due to the lack of data on slag analyses containing less than 2 pct silica, the extension of lines of constant desulfurizing power below this value was considered to be unjustified.

Fig. 6 shows clearly the expected increase in the sulfur distribution ratio as the basicity increases. This figure and Fig. 7, which was prepared from Fig. 6 by plotting the sulfur distribution ratio as a function of the base to acid ratio at various iron oxide concentrations, also show the effect of iron oxide. These figures show that the desulfurizing power of a slag containing a base to acid ratio of less than about 2.4 increases progressively as the iron oxide content increases, but more basic slags result in a decrease in desulfurizing power as the iron oxide concentration is increased.

The results presented in Fig. 6 agree fairly well with the results of Grant and Chipman. They determined the correlation between the sulfur distribution ratio and excess base and found that the iron oxide content is without effect on the distribution ratio. A plot of the excess base on a ternary composition diagram reveals that the lines of constant excess base are essentially parallel to the line joining the orthosilicate composition with $(FeO)_t$. A careful study of Fig. 6 shows that for the more basic slags the lines of constant sulfur distribution ratio are practically parallel to the orthosilicate-$(FeO)_t$ join. However, the deviation of the slope of the lines of constant excess base and constant sulfur distribution ratio increases progressively as the slag basicity decreases, especially for slags less basic than the orthosilicate composition. For the case of an acid slag of a composition corresponding to an excess base value of −0.01, the sulfur distribution ratio only increases from 0.7 at 10 mol pct $(FeO)_t$ to 1.3 at 50 mol pct $(FeO)_t$. Since this amount of variation in the sulfur distribution ratio is less than the overall scatter of the points as plotted by Grant and Chipman, it can readily be seen that it would be difficult to establish the slight effect of iron oxide on desulfurization when excess base is used as a parameter.

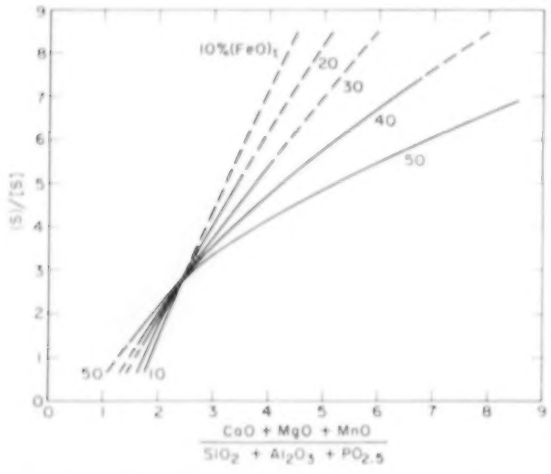

Fig. 7—Desulfurization ratio vs the molar ratio of bases to acids at various molar iron oxide concentrations in the temperature range 1570° to 1630°C.

Effect of Temperature—The effect of temperature on the sulfur distribution ratio was studied by dividing the data into three groups according to the test temperature. The points falling in the temperature range 1570° to 1630°C were used for plotting the curves in Fig. 6. The number of points below 1570°C were insufficient to permit a generalized conclusion regarding the effect of low temperatures, but the points corresponding to temperatures greater than 1630°C indicated that the desulfurizing power of slag decreases slightly as the temperature increases. This temperature effect has been noted previously by others.

Sulfur and Oxygen Equilibria

The relation between the sulfur distribution ratio, as defined previously, and the oxygen in the system can be represented by the reaction

$$O^{2-} + \underline{S} = \underline{O} + S^{2-}. \qquad [2]$$

The equilibrium constant for this reaction can be expressed as follows

$$K_1 = \frac{(a_{s^{2-}})\ [a_O]}{(a_{O^{2-}})\ [a_S]} \qquad [3]$$

where the symbols in parentheses and brackets represent the activities of the elements in the slag and metal, respectively.

The wt pct S in the slag and wt pct S and O in the metal were used in the following calculations. When the latter are substituted for activities in the metal phase, Eq. 3 may be written

$$\frac{K'_1\,(a_{O^{2-}})}{(f_{s^{2-}})} = \frac{(S)\ [O]}{[S]} \qquad [4]$$

where K'_1 is an arbitrary constant, $f_{s^{2-}}$ is the activity coefficient of the sulfide ion, and $a_{O^{2-}}$ is the oxide ion activity in the liquid slag.

The significance of Eq. 4 can be seen from the plot presented in Fig. 8. In this figure the data of Fetters and Chipman[1] were used to plot the ratio $\frac{(S)}{[S]} \cdot [O]$ against the mol pct SiO_2. Grant and Chipman[4] reanalyzed some of the tests for sulfur. In these cases their values are used instead of the original determinations. To minimize the effect of temperature, the oxygen in the metal was corrected to the value in equilibrium with slag at 1600°, and only the data of slags taken at temperatures less

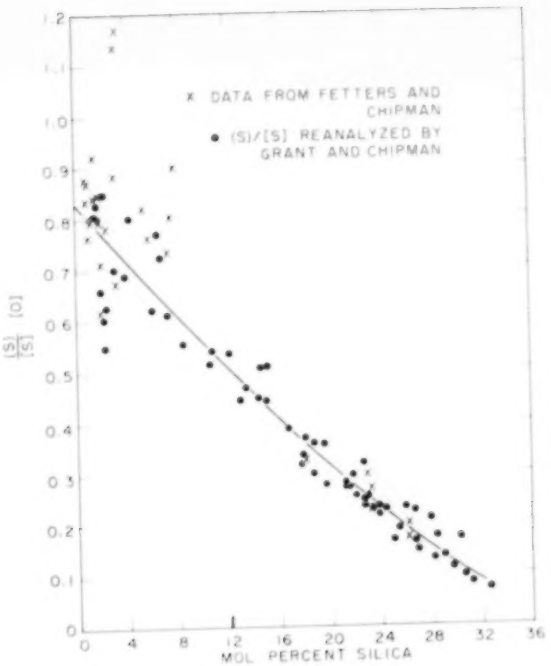

Fig. 8—The ratio $\frac{(S)}{[S]} \cdot [O]$ at 1600°C vs the mol pct silica (slag-metal equilibria data).

than 1700°C were included.

The value of the ratio $\frac{(S)}{[S]} \cdot [O]$, hereafter referred to as the S-O ratio, is in agreement with other work for 0 pct SiO_2. As determined by Bardenheuer and Geller,[18] the sulfur distribution ratio for pure FeO slags in equilibrium with molten iron is 3.6. This corresponds to a value of 0.83 for the S-O ratio when using the oxygen solubility data of Taylor and Chipman.[2]

The data selected for studying the S-O equilibria were not restricted to slags of any given molar ratio of lime to iron oxide, yet there is good correlation between the S-O ratio and the mol pct SiO_2 for all of the slag compositions studied, except those with low silica concentrations, where more scatter is noted. The generally good correlation of the S-O ratio with mol pct SiO_2 and the relationship expressed in Eq. 4 indicate that the ratio of the oxide ion activity to the activity coefficient of the sulfide ion is independent of the molar ratio of lime to iron oxide.

The influence of various factors on reducing the amount of scatter in the points shown in Fig. 8 was investigated. A careful study of the iron oxide and magnesia concentrations was carried out, but the points of high and low concentrations were distributed uniformly about the curved line in both cases. The trivalent iron was also considered following the treatment of Grant and Chipman,[5] in which the Fe_2O_3 is considered to require one half the number of moles of base as silica for neutralization. The plot of the S-O ratio vs the mol pct (SiO_2 + ½ Fe_2O_3) resulted only in a very slight shift in the position of the curve to the right of that shown in Fig. 8, without significantly decreasing scatter.

The results presented in Fig. 8 may be compared with the results of St. Pierre[19] on the sulfur content of iron oxide slags at 1550°C in equilibrium with an atmosphere obtained from an inlet gas mixture in which $SO_2/CO = 34.1$. The range of slag compositions included pure iron oxide slags, iron oxide slags containing lime up to saturation, and lime-silica-iron oxide slags with a molar ratio of lime to silica of 2.24, 1.28, and 0.52.

In order to compare the data on the same basis, the free energy data for the reactions

$$\tfrac{1}{2}\, S_2(g) = S \qquad [5]$$

$$\tfrac{1}{2}\, O_2(g) = O \qquad [6]$$

given in *Basic Open Hearth Steelmaking*[17] were used to caluculate the ratio $\frac{(S)}{[S]} \cdot [O]$ at 1550°C. It is not suggested that molten iron can coexist with an atmosphere $SO_2/CO = 34.1$. This calculation merely provides a basis for comparing the two sets of data.

The similarity between the results presented in Fig. 8 and those presented in Fig. 9, based on St. Pierre's data, is readily noted. The values of the S-O ratio as calculated from the slag-gas equilibria data are slightly higher than for the slag-metal data, but otherwise the two curves are in good agreement. The points representing slags having V-ratios of 2.24, 1.28, and 0.52 in Fig. 9 all lie along the same line. The iron oxide content was less than 2 pct in two of the slag samples. This indicates that the S-O ratio is not dependent on the molar ratio of lime to iron oxide over the range of slag compositions studied. An exception to this, not indicated in Fig. 9, was found in the iron oxide-lime slags. The addition of lime up to lime saturation in pure iron oxide slags resulted in about a 10 pct decrease in the value of the S-O ratio.

Richardson and Fincham[11] studied sulfur equilibrium between gases containing sulfur and oxygen and liquid silicate and aluminate slags. Their results show that the equilibria between sulfur and oxygen at oxygen pressures less than about 10^{-6} atm can be expressed by the following reaction

$$(O)_{slag} + \tfrac{1}{2}\, S_2(g) = \tfrac{1}{2}\, O_2(g) + (S)_{slag} \qquad [7]$$

They have expressed the capacity of the slag to hold sulfur as the sulfur capacity, $C_s = (S)\ (P_{O_2})^{1/2}/(P_{S_2})^{1/2}$, where (S) is the wt pct S in the slag, and

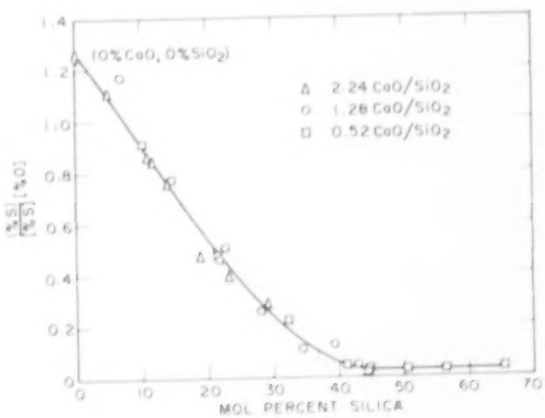

Fig. 9—The ratio $\frac{(S)}{[S]} \cdot [O]$ calculated from gas-slag equilibria data at 1550°C vs the mol pct silica in the slag system CaO-SiO_2-(FeO).

P_{O_2} and P_{S_2} are the partial pressures of oxygen and sulfur, respectively, resulting from heating gas mixtures to elevated temperatures. They have studied the sulfur capacity of various slag compositions of the CaO-SiO_2-Al_2O_3 system and the $(FeO)_t$-SiO_2 system. Their results can be compared with Fig. 9 by using the free energy data for the reactions given in Eqs. 5 and 6. The results of Richardson and Fincham for the $(FeO)_t$-SiO_2 system are in good agreement with Fig. 9. However, their results for the CaO-SiO_2 system give a much lower value for the S-O ratio, which, in view of the higher values determined by St. Pierre for lime-silica slags containing about 2 pct $(FeO)_t$, indicate that the value of the S-O ratio is dependent upon the iron oxide at low concentrations. Additional experimental work is necessary to determine the effect of iron oxide at low concentrations.

Relationships Among Slag Composition, Sulfur Distribution Ratio, and Carbon and Oxygen Content of the Metal

The equilibrium oxygen and carbon contents of the metal at 1600°C and the distribution of sulfur between slag and metal are shown in Fig. 10 as a function of the slag composition. The broken and solid curves represent lines of constant molar ratio (pct CaO + pct MgO)/pct SiO_2 and mol pct iron oxide, respectively. The equilibrium oxygen content of the metal at 1600°C was determined from the iron oxide activity curves shown in Fig. 1, with the exception of the estimated values for slags containing 10 mol pct iron oxide and having basicity values of less than 3.5. The limiting carbon content of the metal at 1600°C was determined from the equilibrium oxygen content of the metal, assuming a carbon monoxide pressure of 1.5 atm, as mentioned previously. The sulfur distribution ratio was then calculated by using the equilibrium oxygen content and the plot of the S-O ratio shown in Fig. 8.

Fig. 10 also shows that, as the basicity increases to about 1.5 to 2.0 at a constant iron oxide concentration, the limiting carbon content decreases slightly, while at the same time the sulfur distribution increases steeply. At the same iron oxide content, a further increase in the basicity results in a still greater increase in the sulfur distribution ratio as well as an increase in the equilibrium carbon content. These results may not be attained in the open hearth, but they may have utility in determining the proper additions required for slag control.

The distribution of sulfur between metal and simple slag systems as shown in Fig. 10 is in good agreement with the results shown in Fig. 7 for the more complex slag systems. There is a slight difference between the two figures for a basicity ratio of about 2.5, but in general it is not significant.

Summary

It is hoped that the combined results which have been prepared from several independent studies of the equilibrium between molten iron and simple basic and acid slags will be a useful contribution to our general knowledge of some of the fundamental chemical reactions which are important in the refining of steel in the open hearth. The results presented in this paper clearly show the effects of basicity and iron oxide concentration of the slag on the distribution of sulfur between slag and metal and on the equilibrium carbon and oxygen contents of the metal.

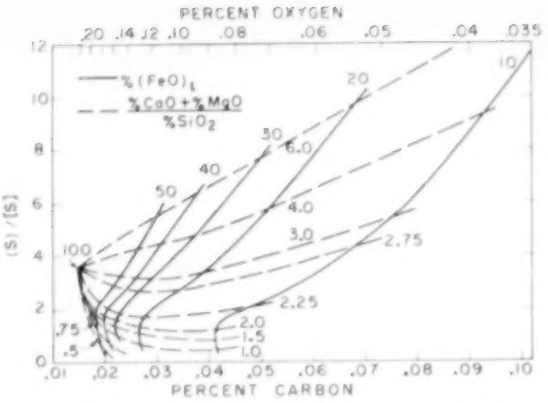

Fig. 10—Relationships among carbon and oxygen contents of the metal, the sulfur ratio, and the slag composition on a mol fraction basis at 1600°C.

In addition, the equilibrium between sulfur and oxygen is presented to show an overall reaction which is important in sulfur control. The results of the sulfur and oxygen equilibria were used to determine the relationship between the limiting carbon content of the metal and the corresponding sulfur distribution ratio. It will be interesting to review practical open hearth data in the light of these results to establish the extent to which equilibrium is reached.

Acknowledgments

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IMD Schedules Annual Meeting in Cleveland

The IMD Annual Fall Meeting is scheduled for October 8-10, 1956 in Cleveland. On the agenda are technical sessions, a cocktail party and dinner. Registration will be at the Carter Hotel and includes a set of available reprints and abstracts. The price for members is $2.00, and $4.00 for non-members. Hotel reservations may be obtained by writing to: Metal Show Housing Bureau, 511 Terminal Tower, Cleveland 13, Ohio.

M. E. Nicholson, IMD programs committee chairman, and W. M. Baldwin, Jr., are arranging the technical sessions. R. F. Hehemann, AIME Cleveland local section chairman, will arrange the social events for the meeting.

Metals Branch Lists Nominations for 1957

Iron and Steel Div.

Chairman: Harold B. Emerick, Jones & Laughlin Steel Corp.; *Vice Chairmen:* John Golden, U. S. Steel Corp.; Karl Fetters, Youngstown Sheet & Tube Co.; Kurt Neustaetter, Inland Steel Co.; *Secretary-Treasurer:* R. W. Shearman, AIME. *Executive Committee* (1957-1960): G. Derge, Carnegie Institute of Technology; Paul E. Lindberg, McLouth Steel Corp.; F. J. McMulkin, Dominion Foundries & Steel.

Extractive Metallurgy Div.

Chairman (Automatic): B. W. Gonser, Battelle Memorial Institute; *Chairman-elect*, H. H. Kellogg, Columbia University; *Secretary*: R. W. Shearman. *Executive Committee* (1957-1960): C. C. Long, St. Joseph Lead Co.; J. H. Schloen, Canadian Copper Refiners Ltd.

Institute of Metals Div.

Chairman: W. R. Hibbard, Jr., General Electric Co.; *Senior Vice Chairman:* W. J. Harris, Jr., Battelle Memorial Institute; *Vice Chairman:* Oscar Marzke, Naval Research Laboratory; *Secretary:* R. W. Shearman, AIME. *Executive Committee* (1957-1960): J. H. Frye, Oak Ridge National Laboratory; H. Y. Hunsicker, Aluminum Co. of America; Donald J. Blickwede, Bethlehem Steel Co.

Metals Meeting in Paris

The Societe Francaise de Metallurgie will hold its Fall Meetings in Paris on October 22-27, 1956. The program containing resumes of the papers to be presented will be distributed in July. All AIME members interested in metallurgical research are cordially invited to participate. Further information may be obtained from the society at 25, Rue de Clichy, Paris (9°).

To Publicize AIME in 1956

Charles Bayer, who successfully handled publicity and news releases for the Annual Meeting last February, has been retained by AIME for the rest of the year. It is planned to issue suitable releases on important AIME meetings, publications, appointments, and awards.

RESEARCH IN PROGRESS SESSION

Institute of Metals Division Fall Meeting

A Research in Progress session is again being organized as a part of the technical program of the AIME Institute of Metals Div. for the forthcoming fall meeting in Cleveland. This session is intended, as in the past, to provide an opportunity for the brief presentation of current research results which are considered by the investigator to be of sufficient interest to warrant public airing before the research is complete or ready for formal publication. All those interested in contributing to this program are invited to send in abstracts (about 200 words) of their proposed presentation to: Prof. B. D. Cullity, Dept. of Metallurgy, University of Notre Dame, Notre Dame, Ind.

The deadline for the receipt of abstracts is set as Sept. 4, 1956. Presentation should be planned for a 5 min time limit plus 5 min for discussion. The abstracts will not be published, but those selected for presentation will be available in mimeographed form at the registration desk during the October meeting.

Contributors are assured that selection will not be on an arbitrary basis nor on the basis of attempts to judge the relative usefulness or value of the papers submitted. Rather, papers will be chosen according to how well the contribution falls into the *Research in Progress* category and according to the possibility of grouping several reports around some unifying principle.

1957 AIME Nominations Slate Announced

For President-Elect

AUGUSTUS B. KINZEL

Augustus B. Kinzel has been nominated for the presidency of AIME in 1958. Vice president of research, Union Carbide & Carbon Corp., N. Y., Dr. Kinzel is a native of New York City and studied at Columbia University, MIT, and the University of Nancy, France, where he received his Doctorate in 1922. He has been AEC chief consultant in metallurgy since 1943, and is a member of the advisory panel on general sciences, Dept. of Defense. Dr. Kinzel, who joined AIME in 1926, has served the Institute as director and was vice president in 1955; he was chosen Howe Memorial Lecturer for 1952. Dr. Kinzel began his career as a metallurgist for General Electric Co., Pittsfield, Mass. in 1919, and joined Union Carbide's research department in 1926. The candidate is a member of the following organizations: ASM, ASTM, SAE, AISI, International Acetylene Assn., and the International Institute of Welding. He has contributed numerous articles to technical publications and is also the author of *Alloys of Iron and Chromium*, volumes 1 and 2.

For Vice Presidents

E. C. Babson, manager, Canadian Div., Union Oil Co. of California, in Calgary, Alberta since 1951, has been nominated as AIME vice president. A native of Los Angeles, he attended grade and high school in Portland, Ore., and then went on to study metallurgy at Stanford University where he received his A.B. in 1930 and M.E. in 1933. Mr. Babson began his career with Union Oil, as a research chemist in Wilmington, Calif., later becoming production engineer. For two years he lectured in metallurgy as part of the war training program at the University of Southern California. Mr. Babson was a partner of Peerless Pacific Co., Portland, until 1948 and then returned to Union Oil in Calif. as staff engineer. A year later he was promoted to assistant chief petroleum engineer. He has been chairman of the Pacific Coast District, American Petroleum Institute, and acting chief, Reserves Section, Production Div., Petroleum Administration for Defense. Mr. Babson is serving as AIME Director through 1957.

E. C. BABSON

Roger V. Pierce, a member of AIME since 1935, has been nominated as vice president for three years. Born in Pacific City, Wash., he attended the Montana School of Mines in Butte, where he obtained his B.S. and M.S. degrees. Mr. Pierce was first employed as miner and shift boss at The Anaconda Co. in 1936. A year later he joined Ingersoll Rand Co. where he worked for ten years as mechanization specialist for Canada, Mexico and the U. S. He has been consultant to such companies as Cerro de Pasco, and San Francisco Chemical Co. Since 1950, Mr. Pierce has been consulting mining engineer and president of Machinery Center Inc. Active in AIME, he was chairman, section delegate and a member of the executive committee of the Utah Section. In addition, Mr. Pierce has headed the technical publications committee, Mining Subdivision, which he also served as program chairman.

R. V. PIERCE

For Directors

Luther C. Campbell, a nominee for the 1957 Board of Directors, is a vice-president of the coal division, Eastern Gas & Fuel Assoc., Pittsburgh, and has served as president of the National Coal Assn. since 1952. Born in Renovo, Pa., he graduated from Pennsylvania State University in 1915 with a degree in mining engineering. A mining official since 1915, first with Rand Mining Co., Nev., he was later general superintendent of coal mining operations for McKinney Steel Co. in Ky. In 1927 he held this position at Melcroft Co., then at Elkhorn Piney Coal Mining Co.; both companies are now part of Eastern Gas & Fuel Assoc., of which he has been vice president since 1943. Mr. Campbell joined AIME in 1935, and from 1946-1948 was a member of its committee for promotion of student interest in coal

John S. Bell, Chairman of the Nominating Committee, has announced nominations for President-Elect, Vice Presidents, and Directors for 1957. As provided in Art. IX, Sec. 2, of the bylaws, 25 Members or Associate Members may sign and transmit to the Secretary's office prior to September 1, 1956 "any complete or partial ticket of nominees," should they wish other candidates to be considered.

In such instance, a letter ballot will be forwarded to all Members in good standing in the United States, Canada, and Mexico, tabulating both the official ballot and any supplementary nominations. If no supplementary nominations are thus received, no letter ballot will be printed, and nominees on the official ballot shall be declared duly elected at the meeting of the Board of Directors or the Executive Committee in November.

mining as a career. He served as chairman of the NCA safety committee and organized their safety division in 1948. For a number of years he has participated actively in the work of Bituminous Coal Research Inc., the American Mining Congress, and the National Safety Council, as well as 11 other regional engineering and mining organizations. Since 1945 Mr. Campbell has been a member of the Old Timers Club, a group of industry leaders dedicated to helping young men attain professional careers in the coal industry.

L. C. CAMPBELL

Thomas C. Frick, chairman of AIME Petroleum Branch 1956, has been nominated director for a three-year term. Past-chairman of East Texas Local Section and of the Permian Basin Section, he is a native of Collinsville, Oklahoma and attended Missouri School of Mines. He transferred to the University of Tulsa, graduating in 1933, with the first class in petroleum engineering. Mr. Frick joined the refining department of Phillips Petroleum Co. where he worked for three years as engineer. He joined AIME in 1936 shortly before returning to the University of Tulsa where he was assistant professor and head of the production department of the School of Petroleum Engineering. In 1941 Mr. Frick accepted a position with Atlantic Refining Co. in Dallas as job analyst, and in 1944 returned to petroleum engineering as district superintendent for the company in East Texas; he was regional manager for Atlantic from 1951-1954 and last year assumed his present post, as manager, Production Div. of the Domestic Crude Oil Production Dept.

T. C. FRICK

F. W. STRANDBERG

Fred W. Strandberg, recently promoted to chief mining engineer at The Anaconda Copper Mining Co., Butte, Mont., has been nominated as an AIME director for a three-year term. Born and raised in Iowa, he joined Anaconda in 1916 after graduating from Iowa State College with a B.S. degree in mining engineering. He has been with that company ever since, except for a brief stint at the Walker Mine in California from 1930-1933. A member of AIME since 1939, Mr. Strandberg was Montana Section delegate from 1950-1955, and chairman of the section in 1953.

Arthur W. Thornton, assistant to the vice president-operations, National Tube Div., U. S. Steel Corp., Pittsburgh, has been nominated to serve as an AIME director for a three-year term. A member of the Institute since 1947, he has served on the executive committee of the NOHC,

A. W. THORNTON

the Bessemer Committee, and was ISD chairman. He is now chairman of the Metals Branch. Born in McKeesport, Pa., Mr. Thornton graduated from Lehigh University in 1931 with a degree in metallurgical engineering. He was affiliated with Weirton Steel Co. in W. Va., and was a member of the metallurgical department of Duquesne Works of U. S. Steel. Since 1934 he has been with the National Tube Div., U. S. Steel, beginning as steel works metallurgist, then superintendent of the Open Hearth and Bessemer Dept., later assistant general superintendent, attaining his current post in 1954.

James S. Vanick has been nominated as director for a three-year term. His lengthy association with the Institute began in 1921; he has since served AIME as chairman of the New York Section. A research metallurgist for International Nickel Co., N. Y., since 1922, Mr. Vanick is the recipient of the Wm. H. McFadden Gold Medal for 1956, of the American Foundrymen's Society.

J. S. VANICK

Born in Cleveland, he received his basic engineering training there, at Case Institute of Technology and continued at George Washington University where he obtained his Master's Degree. Mr. Vanick is a member of AFS, ASME, ASM, ASTM, Corrosion Engineers, and the British Iron & Steel Institute, serving these groups as chairman of various committees. He has written over 50 papers for technical meetings and for the trade press, on ordnance materials, cast metals production and corrosion effects.

Lamar Weaver, general superintendent, Tennessee Copper Co., has been nominated an AIME director for a three-year term. Born in Palmetto, Georgia, Mr. Weaver studied at N. Georgia Agricultural College, where he received his B.S in mining engineering. In 1922 he joined Tennessee Copper in Copperhill, Tenn. as a mine engineer, and six years later became assistant superintendent of mines. By 1938 he was superintendent, and in 1953 assumed his present post. A member of AIME since 1938, Mr. Weaver was chairman of the Southeastern Section in 1952 and has been section delegate since 1953.

LAMAR WEAVER

Around the Sections

• The **New York** Section held a Hob-Nob meeting on May 24 at the Brass Rail Restaurant in association with the powder metallurgists in this area and the New York Physical Metallurgy Group.

Featured speaker was N. J. Grant, professor of metallurgy, Massachusetts Institute of Technology. Mr. Grant's subject was "Dispersed Phase Hardening." He reviewed the development and performance of the aluminum and aluminum-oxide alloys of the SAP variety, particularly the theory behind high strength values achieved. Mr.Grant also described the problems to be solved before successful alloys of the metal-metal oxide type become practical.

• The **Washington, D. C., Section** made its annual field trip on June 5. This year arrangements were made by J. C. Williamson for members to tour the Halethorpe plant of Kaiser Aluminum & Chemical Corp. in Baltimore. After the tour a buffet luncheon was served by the company at Friendship International Airport.

• AIME President, Carl E. Reistle, Jr., visited the **Colorado Plateau Section** at Grand Junction, Colo., on May 22, accompanied by R. E. O'Brien, AIME Field Secretary. After attending an informal luncheon with officers and directors of the Section, and representatives of the uranium industry, Mr. Reistle visited the sampling and assaying installations of Lucius Pitkin Inc., and the pilot plant installation of National Lead Co. at the U. S. Atomic Energy Commission compound. In the evening Mr. Reistle addressed the 60 members and guests at a dinner given in his honor at the Redlands Club.

• The **El Paso Metals Section** met on May 24 at the Hotel Cortez. The principal speaker was AIME President, Carl E. Reistle, Jr.

• The **Montana Section** held a joint meeting on May 26 with the Montana Society of Engineers at Anaconda. F. W. Beichley, regional engineer and service supervisor, Westinghouse Corp., gave a talk entitled "Research—Industries Greatest Adventure." The technical session was preceded by an inspection tour of the Anaconda Reduction Works arranged by R. G. Bowman.

• AIME President, Carl E. Reistle, Jr., was the guest of honor at a dinner-meeting held jointly by the **Colorado Section** and the **Denver Petroleum Section** on May 21 at the University Club, Denver. During the meeting Mr. Reistle was presented with the Denver Dollar Award by Clayton Hill, manager of revenue, City of Denver, in behalf of the mayor and people of Denver. This very singular award was given to Mr. Reistle in recognition of his stature in the minerals industry and as a symbol of the appreciation and hospitality which the City of Denver extends to the oil industry.

The Student Story

(by a member of the St. Louis Local Section)

Realizing that students in the mineral industries are our source of engineers, management personnel, and our big hope for future membership, this section is striving to assist them in their studies, contacts, and finances.

The St. Louis Section of the AIME has under its wing, the student chapters of six schools. These are at St. Louis University; Washington University, St. Louis; University of Missouri, Columbia, Mo.; Missouri School of Mines and Metallurgy, Rolla, Mo.; University of Illinois, Urbana; and Iowa State College, Ames, Iowa.

In the past we have held a special annual program, Student's Conclave, which failed to attract enough regular members. This year we are coaxing students to attend our regular sessions by paying half the cost of their dinner. The effect has been a small but encouraging trickle of students, chiefly from our two schools in St. Louis.

To help those from more distant locations, we invite students from each school to attend one meeting during the year with all expenses paid, including transportation. This results in fairly good attendance by student members.

It seems that a standing offer of a free meal and about $2 per student per 100 miles distance, would afford sufficient financial support to those wishing to come. We believe the habit of attending meetings should be established early. Funds for this endeavor should come from those who will benefit by this association, such as companies who work with minerals and their products, and our Institute, which will provide itself with a new source of interested, *active* members.

Those who should pay, are the companies and the local sections. The companies come first, because they may be able to secure money more easily, and they also benefit the most. Then too, it should be a deductible expense. In fact the companies in the area served by each local section, and by each school which the local section serves, ought properly to contribute to a special fund for this purpose.

In the St. Louis Section we do have such a fund, managed by the local executive committee. It might prove more flexible if handled by a subcommittee devoted to improving relations with student associates. One or two members might be elected or appointed to the executive committee, to function on this subcommittee.

This year serious consideration was given to underwriting the expenses of students wishing to attend the national meeting. One of the difficulties was the selection of students, and therefore, this part of the program did not get under way. Probably if we put up $300 annually, it could be divided among the schools and distributed equally among all the students making the trip.

The choice of students could be left to the student chapters. Although there is no intention of making this help a reward for scholarship, the faculty should assure us that students selected have maintained adequate grades.

Regarding prizes, no section is rich enough to endorse large scholarships although some do assist promising students with small cash awards. Our section, as yet, has no program of this kind. One of our former members, J. H. Steinmesch, gave cash prizes for the three best papers pertaining to the mineral industries. Since his death in 1954, this award, named for the original sponsor, has been made through the generosity of Robert Gill Montgomery. If the standard of excellence continues to be as high as it has been, the award might well become one of the most coveted marks of professional attainment for students. Our Institute, through its local sections, should guarantee the future of awards once they have been started, since they so obviously benefit the profession.

Although we could not organize any help this year, we are proud to announce that three students from the Missouri School of Mines, attended the Annual Meeting, February, in N. Y. and have received several job offers from people they met there. Welcome to Harlan Kebel, Harold Staves, and Michael Vallez, all mining engineers.—*Professor Bob.*

NYU Schedules Titanium Lectures for September

The second annual titanium lecture program, designed especially for practicing engineers, will be held at New York University College of Engineering, Sept. 10-14. It is sponsored jointly by the metallurgical engineering department and N.Y.U.'s office of special services to business and industry.

Titanium authorities from industry and research laboratories will join N.Y.U. faculty members to present 25 lectures. The roster includes two British representatives, Alan D. McQuillan, University of Birmingham and his wife, Marian McQuillan, of Imperial Chemical Industries.

The following subjects will be covered: extraction and melting; phase diagrams, metallography and alloying; heat treatment and mechanical properties, mechanical metallurgy and applications; and fabrication. The lectures, to be given at the University Heights campus in the Bronx, will be supplemented by evening discussion sessions.

Dormitory space will be available from September 9-15. Attendance is limited and those planning to attend may register until August 15. Information and applications can be obtained by writing to Dr. Harold Margolin, N. Y. U. College of Engineering, University Heights 53, N. Y., or by phoning LUdlow 3-6310.

Bessemer Committee Chooses New Name

The AIME Bessemer Steel Committee has officially changed its name to *AIME Converter Steel Committee*. This was decided at the Centennial Celebration held on April 25, at the Lorain, Ohio Works, National Tube Div., U. S. Steel Corp. On p. 741, June JOURNAL OF METALS, the new name was erroneously listed as Pneumatic Converter Committee.

Wins AIME Scholarship

The Leo F. Reinartz Scholarship for a year's study at Carnegie Institute of Technology has been presented by AIME to Larry E. Campbell of Pittsburgh. Valued at $700, the award is given yearly to the high school science student, interested in metallurgy, receiving the highest grade in the NOHC test.

Next Annual Meeting in New Orleans

At the Annual Meeting in New Orleans, Feb. 24-28, 1957, the Mining and Petroleum Branches will make their headquarters in the Roosevelt and the Metals Branch in the Jung Hotel. Tuesday afternoon from 2:15-4:15 will be devoted to an all-Institute program without conflicts. The session will include the Institute's Annual Business Meeting, a review of the year by the President, and a talk by a prominent speaker on a subject of importance and general interest.

The Executive Committee has also voted that no division can hold concurrent sessions. However, the three subdivisions of the mining, geology and geophysics division may hold simultaneous sessions. Thus, the maximum number of sessions that any division, except MGG, can hold will be seven. No Friday sessions may be scheduled. Should more papers be accepted than can be scheduled, the overflow will be directed to divisional, branch, or regional meetings at other times and places. As yet no post-convention trips have been planned. It is felt that Lima, Peru is too distant to be considered.

Pittsburgh AIME—NOHC To Meet in November

The Eleventh Annual Off-the-Record Meeting of the Pittsburgh Sections of AIME and NOHC will be held at the William Penn Hotel in Pittsburgh, on November 2, 1956.

Sessions will be held on coal, petroleum and gas, physical metallurgy, open hearth operations, extractive non-ferrous metallurgy and nonmetallic minerals. There will be an informal luncheon, and a cocktail party and banquet in the evening.

Those wishing copies of the program and registration forms may send their request to Shadburn Marshall, Secretary, Pittsburgh Section, AIME Atomic Power Div., Westinghouse Electric Corp., Bettis Field, Pittsburgh.

Incentive Offered For Securing New Members

An added incentive for securing new members was authorized by the executive committee at their meeting on April 19. The names of all AIME members who obtain five or more new Institute members, will be placed in a hat and five names drawn. The first will receive five years of paid-up AIME dues and the other four, one year of paid-up dues each. This policy has been in effect in the Petroleum Branch and is now extended to the other two branches as well. Until now, AIME has presented a blue and white membership award pin to any member instrumental in securing five new recruits.

New Subsection Organized in Arizona Section

A new subsection of the Arizona Section has been organized, to include members residing in Yavapai County, of which Prescott is the county seat. John W. Still is Chairman; J. D. Lowell, vice chairman; Mark Gemmill, secretary-treasurer; and Fred Gibbs and K. L. Erickson, additional directors.

Sept. 15, 1956 is the deadline for Institute of Metals Div. papers (4 copies) for the 1957 New Orleans Annual Meeting and the deadline for all Iron & Steel Div. and Extractive Metallurgy Div. papers (3 copies) that are to be considered for preprinting. Papers received by this deadline but requiring revision may not be processed in time to permit scheduling for this meeting.

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Personals

Joseph A. Valentin was appointed vice president and manager of sales, Michigan Limestone Div., U.S. Steel Corp. He was formerly manager of the company's northern district. **Carl G. Hogberg** was named assistant vice president. Mr. Valentin was born in Hammond, Mich., and attended Ferris Institute, Big Rapids, Mich., where he received a degree in pharmacy and chemistry. He began his 38 years of service with U.S. Steel in 1918. He was promoted successively to chief chemist, director of industrial relations and plant manager. A native of Escanaba, Mich., Mr. Hogberg joined U.S. Steel in 1935 as a blast furnace apprentice, South Chicago Works.

Michael I. Bricksin has joined the liquid rocket plant of Aerojet General Corp. in Sacramento, Calif., where he will be employed as an engineer.

Edwin S. Bartlett, formerly research metallurgist with Hoskins Mfg. Co., Detroit, has accepted a post as principal metallurgist, Battelle Memorial Institute, Columbus, Ohio.

L. R. COOPER

Lloyd R. Cooper has been promoted from assistant to chief metallurgical engineer, Heppenstall Co. in Pittsburgh. He will coordinate the company's metallurgical departments in Pittsburgh, Indianapolis, and Bridgeport, Conn. Mr. Cooper was with U. S. Steel Corp. from 1936 to 1942 when he joined Heppenstall. After serving as research metallurgist, he was named acting research director in 1946, assuming the directorship in 1948. A graduate of Lehigh University, Mr. Cooper received his chemical engineering degree in 1935. He is a member of the American Chemical Society, ASM, AIME, and ASTM.

R. S. A. Dougherty, manager of research for Bethlehem Steel Co. for the last thirty years, has retired. He will be succeeded by **J. S. Marsh,** former assistant manager of research. **J. E. Eberhardt** has been appointed an assistant manager of the department. A native of San Francisco, Mr. Dougherty is a West Point graduate and was a colonel in the Army Engineer Corps. He joined Bethlehem Steel in 1919, was appointed manager of patents the following year, and in 1926 became manager of research.

J. S. MARSH

Mr. Marsh who hails from Pittsburgh, graduated from Penn State University. In 1930 he became metallurgist for the Engineering Foundation, N.Y. He joined Bethlehem Steel in 1942. The author of several technical works and papers, Mr. Marsh was awarded the AIME Hunt Medal and was the Institute's Howe Memorial lecturer in 1955.

Maurice A. E. Mawby has been awarded the Australasian Institute of Mining & Metallurgy Medal for 1955 in recognition of his contribution to exploration and to nonferrous metallurgy, and also of his continuous public service in many directions associated with mining and metallurgy. Mr. Mawby is director of exploration and research, Consolidated Pty. Ltd., Melbourne, Australia. The medal will be presented to Mr. Mawby during the annual conference of the Institute to be held in August at Broken Hill, New South Wales.

Arthur F. Steeves has joined Westinghouse Electric Corp.'s atomic power division, Pittsburgh, in an engineering capacity. He was formerly employed as a materials engineer in the electronics boat division, General Dynamics Corp., Groton, Conn.

James C. Hodge is now vice president, Warner & Swasey Co., Cleveland. Prior to joining Warner, he was executive vice president of Wellman Engineering Co., Cleveland.

Charles H. Deane has joined the metal department of Bache & Co., N. Y. He will specialize in nonferrous metals.

William J. MacKenzie formerly metallurgical engineer, Western Bar Sales, Youngstown Sheet & Tube Co., Chicago, has retired and is residing in Lake Forest, Ill.

Asoka Gupta, formerly development and research chemist for Burmah Oil Co. Ltd. in Digboi Assam, India, has joined the Indian Iron & Steel Co. Ltd. in W. Bengal, India. Dr. Gupta will work in the company's blast furnace department.

Tadao Tamura has been promoted to director of the research laboratory, Furukawa Electric Co. Ltd., Tokyo.

F. L. Stubbs has moved from Jacksonville, Fla. to Coolangatta, Queensland, Australia. Mr. Stubbs is a graduate of the Colorado School of Mines, Golden, Colo.

A. T. Cape has joined Coast Metals Inc., Little Ferry, N. J., as a metallurgist. He was formerly associated with Raymond Equipment Co., Glendale, Calif.

Harold C. Templeton, has left the Lebanon Steel Foundry, Lebanon, Pa. where he has been chief metallurgist, to join the L.F.M. Co. in Atchison, Kan.

Max Stern, consulting engineer, has moved his offices to 10 E. 60th St., N.Y.C.

Robert C. Rasmussen is now associated with Vanadium Corp. of America, in Cambridge, Ohio. Prior to accepting his new post, Mr. Rasmussen worked for the Pittsburgh Metallurgical Co. Inc., Charleston, S. C.

Kenneth C. Vincent, formerly associated with Armour Research Foundation, Chicago, is now employed at Westvaco Mineral Products Div., in Newark, Calif.

Carlton G. Lutts has retired from his post with the Boston Naval Shipyard, Boston where he was in charge of the materials laboratory. Mr. Lutts is now living on Cabot Farm in Salem, Mass.

Ray P. Dunn is now director of metallurgy, U.S. Reduction Co., East Chicago, Ind. Prior to accepting this post, Mr. Dunn was plant metallurgist for the Doehler-Jarvis Corp., in Chicago.

Samuel S. Shapiro has joined the nuclear division of the Glenn L. Martin Co. in Baltimore. Previously Mr. Shapiro worked in the atomic energy division, Sylvania Electric Products Co., N. Y.

John T. Sherman is assistant director for domestic procurement, Div. of Raw Materials, U. S. Atomic Energy Commission, Washington, D. C. Mr. Sherman was formerly associated with Chemical Construction Corp., N. Y. C.

James T. Semple, who completed his service with the U. S. Armed Forces, is now junior production engineer with Electro Metallurgical Co. which manufactures titanium in Ashtabula, Ohio.

Philip A. Beeson has left American Smelting & Refining Co., Inglewood, Calif., where he was a metallurgical engineer, to join North American Aviation Inc., Los Angeles, as research engineer.

Joseph Babyack of Carnegie Institute of Technology was awarded the AIME Lewis E. Young Award in Pittsburgh on April 11. The award, for good scholarship and professional promise, is named in memory of the first president of the AIME Pittsburgh Section. It is presented each year by the Women's Auxiliary and is accompanied by a $200 cash prize.

Glenn D. Aase is now associated with El Paso Smelting Works, El Paso, Texas. He was a research engineer with American Smelting & Refining Co., Selby, Calif.

W. B. Pierce has been promoted to vice president-sales, Allegheny Ludlum Steel Corp., Pittsburgh. He was formerly technical director for the company in Brackenridge, Pa.

Joao G. Haenel has been elected a director of Companhia Siderurgica Paulista in Brazil. This company will erect an integrated steel mill at Piassaguera near Sao Paulo.

Donald R. Mash is now head of the metallurgy section, atomic energy division, American Standard Co., Redwood City, Calif. Prior to accepting this post, he was metallurgical engineer at California Research & Development Co., Livermore, Calif.

Elmer H. Haug, formerly plant engineer, American Smelting & Refining Co., Selby, Calif., has joined the Southern Peru Copper Corp., in Denver.

William H. Miller has joined Johnson Bronze Co., New Castle, Pa. as a metallurgical engineer.

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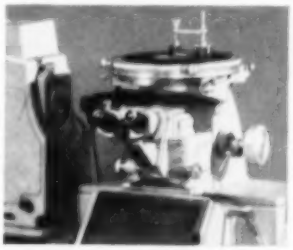

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Rip Van Thompson, formerly superintendent, Metcalf Mining & Metallurgical Co. in Deltah, Utah, is now associated with Samical Minerals Corp., Kingman, Ariz.

R. V. THOMPSON

Carl F. Floe has been elected a director of Walworth Co., N.Y.C., manufacturers of valves and pipe fittings. Dr. Floe is professor of metallurgy and assistant provost, MIT, and has served industry for many years as a consultant in the field of surface hardening of steel. He is a member of ASTM, and ASM and a director of the New England Materials Laboratory.

Charles B. Reed has resigned as chief engineer for Newmont Oil Co., to engage in general consulting practice with headquarters in Austin, Texas. Mr. Reed who was supervising seismologist for Shell Oil Co. from 1936 to 1941, is a member of AIME, SEG, AAPG, AGU and GSA. His biography will appear in the monthly supplement to *Who's Who.*

W. M. Billinghurst has accepted a post as chemical engineer with the Southland Co-operative Phosphate Co. Ltd., Invercargill, New Zealand.

Robert T. Howard is now associated with the James W. Weldon Laboratory, Kansas City. Formerly he was staff engineer, Kansas City Div., Bendix Aviation Corp. The Weldon Laboratory specializes in metallurgical investigations of metals which fail in use, particularly in the transportation field.

Henry L. Kurtz has joined National Forge & Ordnance Co., Irvine, Pa., as assistant chief metallurgist. Previously he was assistant metallurgist, Rod & Wire Div., Youngstown Sheet & Tube Co., Youngstown.

G. J. Brittingham consulting metallurgical engineer has accepted a post with the United Nations Korean Reconstruction Agency with headquarters in Korea. He will remain there for approximately two years.

V. V. Donaldson has joined Alco Products Inc., in Latrobe, Pa. He will work in the research and development section. Mr. Donaldson was metallurgical engineer, Westinghouse Electric Corp., Blairsville, Pa.

Herman S. Rosenbaum is now research asisstant with General Electric Corp., in Schenectady, N. Y. Mr. Rosenbaum, who formerly was associated with the Franklin Institute, Philadelphia, is employed in General Electric's Research Laboratory.

Felice C. Jaffe, formerly with Clark M. Robertson, Milwaukee, is now associated with Newmont Mining Corp., New York.

Ladies' Day In Wyoming Featured Fashion Luncheon

The ladies were not forgotten at the second annual joint meeting of the Casper, Denver, and Billings chapters of AIME in Casper, Wyo., May 19-20. A special fashion luncheon, held in the Gladstone Hotel featured the latest modes modeled by the distaff of Casper. Other social events included a cocktail hour on Friday and the convention dinner that evening.

At the technical sessions on Saturday morning and afternoon, the following papers were presented: *An Atom Smasher for Well Logging,* by Philip W. Martin; and *Water Flood Operations at North Meadow Creek Sussex Sand Unit,* by L. B. Myers.

Results of AIME Membership Survey

A review of the membership applications of 296 members of full professional grade selected at random, (slightly over 2 pct of AIME members) showed that 224, or 75.7 pct, had a college degree; 52, or 17.6 pct, attended college but did not graduate; and 20, or 6.7 pct, never attended college.

Of 113 Associate Members, (slightly over 5 pct of all AIME members in that grade), 75, or 66.3 pct, had a college degree; 25, or 22.1 pct, went to college but received no degree; and 13, or 11.5 pct, did not attend college. Of 153 applications received from Junior Members, (2.8 pct of the total number) only one did not go to college.

New Douglas Medal Committee Chairman

Carleton C. Long has been named new chairman of the Douglas Medal Committee. He will serve in place of Walter A. Dean, who was unable to continue in this capacity this year, due to the pressure of his work. Mr. Long is with St. Joseph Lead Co., Pa.

Electric Furnace Steelmaking Celebrates 50th Anniversary

Pictured above is a 27-year-old elliptical, six electrode furnace. A half-century of electric furnace steelmaking will be feted when the Committee convenes December 5-7 at the Hotel Morrison in Chicago. The group expects to top last year's record attendance of 949 at the scheduled technical sessions and Annual Dinner.

Howard E. Jones is now a partner in the firm of Jones & Mundle Co., manufacturers' representatives, Detroit. He had been district sales representative for the Davis Fire Brick Co., Detroit.

Fred A. Todd has left National Lead Co., Fredericktown, Mo., to accept a post as metallurgical engineer in the process engineering department of C. F. Braun & Co., Alhambra, Calif.

Charles B. Bradfield has joined Schuylkill Products Corp., Baton Rouge, La., as assistant smelter superintendent. Prior to this he held the same post at Dixie Lead Co., Dallas, Texas.

Sam Leber has joined General Electric Co.'s Cleveland wire plant in Euclid, Ohio, as a metallurgist. Prior to this Mr. Leber had been senior research associate, Horizons Inc., Cleveland.

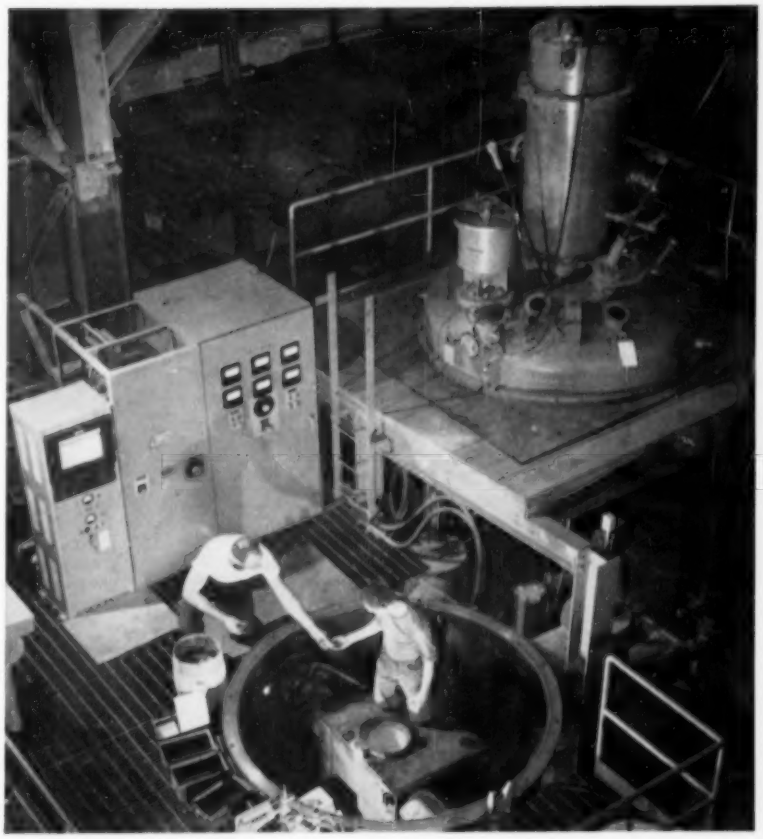
Malcolm F. Parkman has resigned as development engineer, Titanium Metals Corp., Henderson, Nev., to join the Stanford Research Institute. He will be assistant metallurgist at the Institute's headquarters in Menlo Park, Calif.

C. Richard Adelman, Jr., mining and metallurgical engineer, California Research & Development Co., is now project engineer, University of California Radiation Laboratory, located in Livermore, Calif.

Leslie V. Whiton has opened an office in White Plains, N.Y., as consultant on production and marketing of nonferrous metals. He has held various posts including metallurgist and sales manager at Consolidated Mining & Smelting Co. of Canada, Dominion Magnesium Ltd., and Aluminum Co.

L. V. WHITON

of Canada. A graduate of the University of Toronto, Mr. Whiton toured Europe for three months in 1953 studying metal markets. He is a member of ASM, AIME, ASTM and the National Association of Corrosion Engineers.



CVC vacuum furnace at Blairsville, Pa. plant of Westinghouse Electric Corp.

How a vacuum furnace can grow and save money for *Westinghouse*

When Westinghouse wanted a flexible, high-vacuum furnace, they came to us with two special requests:

1. They wanted a furnace with 350 pounds capacity for development work,
2. They wanted to expand this furnace's capacity quickly and economically should the need arise.

Our engineers designed the furnace around a group of modules or "building blocks" to meet these requests.

Westinghouse will up the capacity to 1000 pounds simply by substituting larger crucibles for the 350-pound size now in the furnace. Basically, the rest of the furnace stays the same.

Westinghouse can make the furnace semi-continuous and increase production to 60 tons or more a month of ultra high-purity vacuum metals simply by adding two valved interlocks—one for charging, one for mold removal.

Switching from single to multiple or centrifugal molds requires only a change in the furnace bottom.

In all cases, a minimum number of parts need be purchased.

Three *CVC information memos* outlining the furnace's design and operation in detail are available on request.

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Obituaries

Daniel C. Jackling

An Appreciation by
Louis S. Cates and Erle V. Daveler

To those who had the privilege of working for and with Mr. Jackling—and it was our good fortune to have that privilege for over a quarter of a century—he was affectionately known as "The Boss". He was that—a man among men.

Mr. Jackling was indeed a rare and gifted man. With a vision of a new horizon for copper mining, he possessed the persistent courage, the ability and industry, despite delays, discouragements, and disappointments, to see the vision through and bring it to unbelievable success.

Not the least of the abilities of Mr. Jackling was his inspirational leadership. He himself was a tireless worker, but he possessed the ability to transmit to his co-workers an enthusiasm and devotion to the work at hand so that in heart and soul they were "his team."

Orphaned at the age of two, Mr. Jackling spent his boyhood on Missouri farms with various relatives. He completed his public schooling at sixteen; later, he became attracted to engineering and enrolled at the Missouri School of Mines. A brilliant scholar, he finished the four-year course in three years, graduating in 1892 with a Bachelor of Science degree.

Following graduation, he worked in varying capacities—as miner, mill hand, assayer, metallurgist, and as construction and metallurgical superintendent.

Engaged to examine the copper ore deposit at Bingham Canyon, Utah, he became convinced, after exhaustive testing and planning, that the orebody, despite its low copper content, could be worked profitably. His basic idea, or dream, was that the ore—of a grade far too low to be treated by conventional methods—would become of commercial value if it could be mined and treated by mass means. Accordingly, his revolutionary proposal, made in 1899, was that the waste rock overlying the ore be stripped and the ore thus exposed be mined with steam shovels; that, in place of a 500-ton-per-day concentrator, then regarded as a large concentrator, a 2,000-ton concentrator be constructed. Many experienced mining engineers viewed these plans with skepticism and distrust. Because his ideas were so foreign to the accepted mining practices of the time, financial backing could not at that time be found for the project.

But the dream of developing a mine at the Bingham Canyon deposit persisted. It was not until 1903 however, that others became persuaded, in no small part due to the character of the man, to back the dream and to form the Utah Copper Co. to develop the property.

After some four years spent in preparing and developing an open pit for mining by steam shovels; building not a 2000-ton concentrator, but one of 6000-ton capacity; building a railway system; and providing other production facilities—production was started in August 1907. Attesting to the rightness of "The Boss's" ideas and the care and thought he expended on the plans to bring this new and untried method of copper mining into being, the operation proved successful from the start.

Mr. Jackling has been called the "Father of the Porphyries". The success which attended the development of the low-grade orebody at Bingham Canyon revolutionized copper mining. As a result of Mr. Jackling's pioneering in making low-grade porphyry ores commercial ores, other major low-grade copper deposits in the United States, as well as abroad, have been successfully opened for commercial production. The billions of pounds of copper since produced from porphyry mines in the U. S. and abroad, evidence the tremendous contribution to society made possible by this man's vision and his creative work at Bingham Canyon.

Mr. Jackling received many honors during his long, busy, and useful life. He received from President Woodrow Wilson a Distinguished Service Medal in 1919 for his service in World War I as director of all U. S. Government explosives plants.

Four of the world's highest awards ever accorded an individual engineer were given to him. In 1926, he received the Gold Medal Award of the Mining and Metallurgical Society of America for "distinguished service in the development of low-grade mines." The AIME presented the William Lawrence Saunders Gold Medal for Achievement in Mining to him in 1930. He was awarded the John Fritz Medal of the four Founder Engineering Societies in 1933 for "notable industrial achievement in initiating mass production of copper low grade ores, through application of engineering principles." The Washington Award was presented to him in 1940 in Chicago for "pioneering in large scale mining and treatment of low grade copper ores, releasing vast resources from formerly worthless deposits."

Withal, Mr. Jackling was a sincere and generous leader. In his acceptance speech on the occasion of receiving the Mining and Metallurgical Society of America's Gold Medal in 1926, he said:

". . . I would not be true to myself, to my profession, my associates, or my friends, including all to whom I now speak, if I failed to accord to my co-workers the credit they merit at least equally with me for advancement in the mining practice, both technical and commercial, on account of which this honorary emblem is now bestowed . . . . ."

This typifies the man. In spite of his individual attainments, he would accept no honor without according to his co-workers "the credit they merit at least equally with me." His brilliancy in his chosen profession was recognized by all. To those who knew and worked with him, their great admiration for his high attainments was no less than their deep personal affection and respect for him as a man and a friend.

Memorial Resolution

in honor of

DANIEL COWAN JACKLING

WHEREAS, with the passing of Daniel C. Jackling at his home in Woodside, California, on March 13, 1956, the Institute has lost one of its most loyal members; and

WHEREAS, during his fifty-six years of membership he served the Institute as a Director from 1914 to 1916, 1927 to 1929, and 1938 to 1940, and was President in 1938; and

WHEREAS, he was William Lawrence Saunders Medalist in 1930 for distinguished achievement in mining; John Fritz Medalist in 1933 for notable industrial achievement; and in 1941 was elected to Honorary Membership in the Institute; and

WHEREAS, he showed deep interest in assisting young men to enter our profession and in recognizing their progress and achievements;

THEREFORE, BE IT RESOLVED, That the American Institute of Mining, Metallurgical, and Petroleum Engineers records with deep sorrow the passing of one of America's most distinguished engineers, and one of the Institute's best friends and advocates; and

BE IT FURTHER RESOLVED, That this Resolution be included in the minutes of this meeting and that a copy be sent to Mrs. Jackling.

June 20, 1956 — Board of Directors

Obits Continued

Oliver P. Luetscher (Member 1920) died Jan. 3, 1956. A consulting engineer who specialized in metallurgical furnaces, Mr. Luetscher was born in Pittsburgh in 1886. He received his B.S. in chemical engineering in 1906 from Washington University, St. Louis. Upon graduation he joined United Engineering & Foundry Co., Pittsburgh. In 1915 he became associated with Illinois Zinc Co., Peru, Ill., as chief engineer. At the time of his death he was engaged in private consulting work in Tiskilwa, Ill.

Ronald C. Rutherford (Member 1940) died April 25, 1956, at his home in Los Altos, Calif. A well-known metallurgist, Mr. Rutherford was born in Winnipeg, Manitoba, Canada, on Jan. 3, 1885. Upon graduation from the Royal Technical College of Glasgow, Scotland in 1908, he served as chemist and metallurgist in Canada for International Nickel Co. and Consolidated Mining & Smelting Co. In 1918, he joined the American Smelting & Refining Co. at its East Helena, Montana plant as chief chemist, and in 1921 was promoted to assistant superintendent. From 1927 to 1937 he was successively assistant superintendent of the Lead Dept. at the El Paso, Texas plant and superintendent and manager of American Smelting Co.'s Chihuahua plant in Mexico. From 1937 to 1940 Mr. Rutherford was chief metallurgist of the company's Mexican Smelting Dept. He served as traveling metallurgist and inventory auditor visiting all the company's smelters and refineries from 1941 to the time of his retirement in January 1950. After retirement he received an assignment as consulting engineer to the U. S. Government, which entailed making a survey of various lead mines in Turkey for the E.C.A.

Robert W. Buzzard (Member 1946) died on May 3, 1956, at Ames, Iowa, while attending an AEC metallurgy information conference. He was project leader for the metallurgy division, National Bureau of Standards in Washington, D. C. Born in Fenton, Michigan in 1904, Mr. Buzzard attended Michigan State College where he received his B.S. degree in chemical and metallurgical engineering. After graduation he worked briefly for the Navy Dept. in Detroit, before joining the National Bureau of Standards in 1927. Mr. Buzzard obtained patents for his work on anodic coatings to protect aluminum and magnesium alloys against corrosion. He also published papers on hydrogen embrittlement of steel. During World War II, he served in Africa and Europe and was awarded the Bronze Star. Mr. Buzzard's latest interests were in the development of a unique approach to alloying by means of an ionic analysis method. He was a member of ASM, British Institute for Metals, Army Ordnance Assn., and the Electrochemical Society.

Archer E. Wheeler (Legion of Honor Member 1899) died Mar. 8, 1956. Mr. Wheeler, who retired in 1947, was a consulting metallurgical engineer in New York City. He was born in Auburn, Maine, in 1868. After completing a special three-year course at Massachusetts Institute of Technology, Mr. Wheeler worked for Boston & Montana Consolidated Copper & Silver Mining Co., Great Falls, Mont. Later he joined the firm of Robert Williams & Co. in London, England. In 1917 he returned to New York and went into private practice.

Necrology

Date Elected	Name	Date of Death
1939	L. D. Bartell	Unknown
1946	Robert W. Buzzard	May 3, 1956
1914	Frank T. Donahoe	May 3, 1956
1906	Stanley N. Graham	Feb. 21, 1956
1949	Neal Ham	Mar. 7, 1956
1915	James L. Head	June 3, 1956
1942	John Tate Lewis, Jr.	May 2, 1956
1920	Oliver P. Luetscher	Jan. 3, 1956
1952	J. G. Miller	May 6, 1956
1938	Harry M. Moses	Apr. 1, 1956
1955	P. Ross Pratt	Sept. 15, 1955
1953	Giovanni Puppo	May 30, 1955
1940	R. C. Rutherford	Apr. 25, 1956

Proposed for Membership
— Metals Branch AIME —

Total AIME membership on June 30, 1956 was 24,981; in addition 2,761 Student Associates were enrolled.

ADMISSIONS COMMITTEE

R. B. Caples, Chairman; F. A. Ayer, Vice-Chairman; A. C. Brinker, R. H. Dickson, C. R. Dodson, R. B. Fulton, T. D. Jones, F. W. Hanson, Sidney Rolle, F. T. Sisco, O. B. J. Fraser, F. W. McQuiston, Jr., A. R. Lytle, L. P. Warriner.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

Members

John Billings, Burnaby, B. C., Canada
Edmond J. Brandon, Poland, Ohio
Salvador D. Capocasale, Buenos Aires, Argentina
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Edward R. Frederick, Pittsburgh
Douglas J. Harvey, Utica, Mich.
William B. Hayes, Niagara Falls, N. Y.
James L. Hooper, Columbia Falls, Mont.
Joseph E. Ippoliti, Freeland, Pa.
Frank Jacobs, Jr., Wichita, Kan.
Wilfred A. Jenks, Warrington, Lancashire, England
Roland R. Kiessling, Stockholm, Sweden
Carlyle B. Lambert, Provo, Utah
William E. Littell, Jr., Cleveland
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Hoy O. McIntire, Columbus, Ohio
John D. McIver, Salt Lake City
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Alexander Nagy, Bridgeville, Pa.
Richard G. Nelson, Bridgeville, Pa.
Kendal M. Ogilvie, Mishawaka, Ind.
Virgil F. Ostler, Bountiful, Utah
Peter Rossin, Bridgeville, Pa.
Oscar J. Scherer, Henderson, Nev.
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Arnold Z. Sleeper, Oswego, Ore.
Haral L. Stuart, Westbury, N. Y.
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Jack N. Koch, Seattle
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Uir Drimmer, Chicago
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Daniel J. Hayes, Ann Arbor, Mich.
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Ralph D. Hindson, Hamilton, Ont., Canada
Lester J. Randall, Wallace, Idaho

Junior to Member

Thomas Marshall, Wellington, New Zealand

REINSTATEMENT

Members

Robert G. Abbey, Bala-Cynwyd, Pa.

REINSTATEMENT—CHANGE OF STATUS

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Student to Member

Arthur F. Betchart, Newcastle, N. B., Canada
Val Kudryk, Brooklyn

Junior to Associate

Alden McElrath, Jr., Piedmont, Calif.
Saul Z. Silver, Miami, Fla.

Student to Junior

Donald A. Cogley, Venetta, Pa.
Laurence E. Poteat, Niagara Falls, N. Y.

Joins Editorial Staff

JAMES J. BURKE

The new assistant editor of JOURNAL OF METALS calls Berkely, California, his home town. After attending high school there, Jim received a scholarship to Yale University where he majored in metallurgy. At Yale, Jim was managing editor of the Yale Record, the campus humor magazine, and served a term as president of the AIME student chapter. He also found time to join the Davenport Cricket Club and to listen to his favorite recordings of classical music. Summer vacations were spent working for the American Smelting & Refining Co. at Selby, and the General Metals Corp. foundry in Oakland, Calif. Jim came to 39th Street immediately after graduation in June.

— Coming Events —

Aug. 6-8, SAE, West Coast meeting, Mark Hopkins Hotel, San Francisco.

Aug. 19-24, Combustion Inst., sixth international symposium on combustion, New Haven, Conn.

Aug. 23-25, National Council of State Boards of Engineering Examiners, Hotel Statler, Los Angeles.

Sept. 9-12, AIChE, William Penn Hotel, Pittsburgh.

Sept. 10-14, New York University, second annual titanium lecture program designed for practicing engineers, College of Engineering, NYU, New York.

Sept. 11, AIME, NOHC Chicago Local Section, annual golf party, Midlothian Country Club, Chicago.

Sept. 16-22, ASTM, Pacific Coast meeting, Statler Hotel, Los Angeles.

Sept. 26-28, AIME, Rocky Mountain Minerals Conference, Newhouse Hotel, Salt Lake City.

Sept. 27-28, AIME, NOHC Southwestern Local Section, annual fall get together and plant trip. Sept. 27, trip to Colorado Fuel & Iron Co., Pueblo, Colo. Sept. 28, technical sessions, Broadmoor Hotel, Colorado Springs, Colo.

Oct. 5, AIME, NOHC Southern Ohio Local Section, annual conference, Deshler-Hilton Hotel, Columbus, Ohio.

Oct. 8-10, AIME, IMD Fall Meeting, Hotel Carter, Cleveland.

Oct. 8-12, ASM, national metal congress and exposition, Cleveland Public Auditorium, Cleveland.

Oct. 12, AIME, NOHC Eastern Local Section, annual meeting.

Oct. 14-17, AIME, Petroleum Branch, Biltmore Hotel, Los Angeles.

Oct. 16-18, AIEE and Amer. Physical Soc., conference on magnetism and magnetic materials, Hotel Statler, Boston.

Oct. 22-24, ASA, annual meeting, Hotel Roosevelt, New York.

Oct. 24-25, AIME-ASME, Joint Solid Fuels Conference, Sheraton-Park Hotel, Washington, D. C.

Oct. 28-Nov. 1, Electrochemical Soc., Statler Hotel, Cleveland.

Nov. 2, AIME, Pittsburgh Local Section, NOHC Pittsburgh Local Section, annual off-the-record meeting, William Penn Hotel, Pittsburgh.

Nov. 8-10, AIME, Mining Branch, Northeast Regional Meeting, Hotel Hershey, Hershey, Pa.

Nov. 13, AIME, NOHC Buffalo Local Section, annual meeting.

Dec. 5-7, AIME, Electric Furnace Steel Conference, Hotel Morrison, Chicago.

Dec. 9-12, AIChE, annual meeting, Statler Hotel, Boston.

Feb. 24-28, 1957, AIME Annual Meeting, Roosevelt and Jung Hotels, New Orleans.

Mar. 10-16, 1957, EJC Second Nuclear Engineering and Science Congress, Convention Hall, Philadelphia.

Mar. 25-29, 1957, ASM And Others, Tenth Western Metal Exposition and Congress, Pan-Pacific Auditorium and Ambassador, Los Angeles.

Apr. 8-10, 1957, AIME National Open Hearth Steel and Blast Furnace, Coke Oven, and Raw Materials Conferences, William Penn Hotel, Pittsburgh.

Apr. 11-13, 1957, AIME, Pacific Northwest Regional Conference, Portland, Ore.

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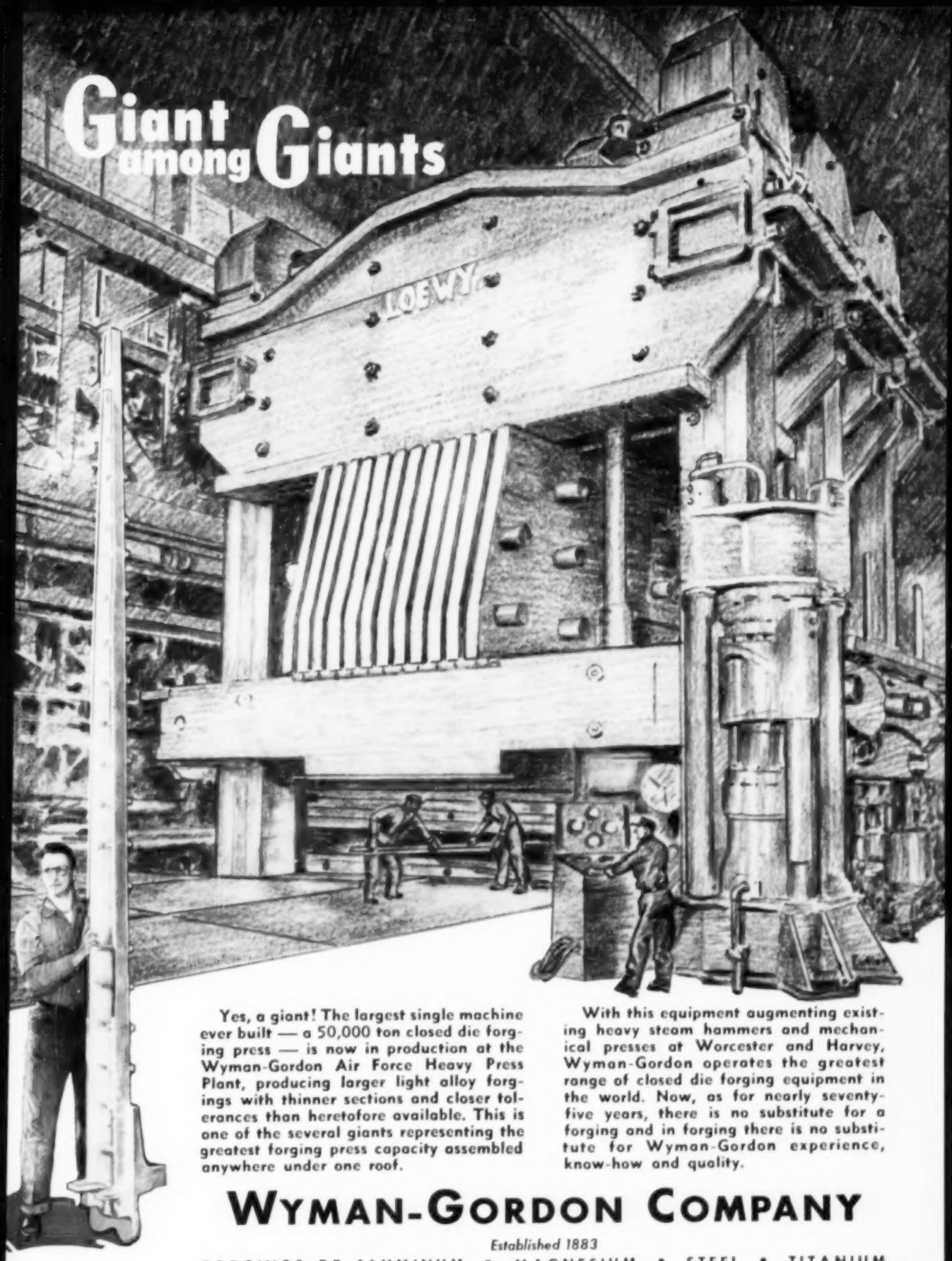

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